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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MANAGEMENT OF LABOR IN SUCCESSFUL METAL-MINE OPERATIONS[1]

By Charles Will Wright[2]

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INTRODUCTION

This paper is one of a series dealing with mining problems and summarizing the data contained in the Information Circulars on individual mines that have been prepared by the managers or superintendents and published by the Bureau of Mines.

The purpose of this paper is to present the problems encountered in the development of an efficient mine organization and to show the means used by certain mining companies in promoting efficiency. There is also a discussion of the advantages and disadvantages of the various systems of paying and handling labor and a description of their application at the individual mines.

The manager of every producing mine, with the exception of gold mines, is to-day faced with the necessity of lowering unit production costs to meet the present low market prices for the metal products. To do so he is introducing ways to improve the efficiency of labor and where possible is maintaining his operative force and the wage scale.

At those mines where production has been suspended, there are certain fixed expenditures for the maintenance of both the mine workings and organization, as well as taxes that must continue in order to keep the mine alive, and just how far the management can go in making such expenditures is another problem that the management has to solve. At certain mines,

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6650."

2 - Chief engineer, mining division, U. S. Bureau of Mines.

however, it has been possible to reduce production costs by one-half, due to improved efficiency and perhaps a small decrease in wages, which reduction has permitted these mines to continue profitable operations. It has also been noted that where workmen have been dropped from the company's pay roll, there has been a greater output per man among those remaining because of fear of losing their jobs.

As over 60 per cent of mining costs are for labor, the most vital problem in mine management is the reduction of labor cost per unit of output, or in other words, the increase in human productiveness.

The success of many mining operations depends upon the attitude of the workmen toward their employer and their willingness to work well. The importance of such an attitude is evident. Under present conditions there are perhaps few places filled by unwilling workers.

The following table shows primarily the percentage of underground mining costs for labor classified according to the mining method used; the man-hours per ton mined have been added for purposes of comparison:

	Number of mines	Mining cost for labor per cent of total[1]	Number of mines[2]	Man-hours per ton mined[2]
Square set..........	12	65.4	61	5.15
Cut-and-fill	8	62.2	21	4.34
Shrinkage...........	16	61.7	37	1.33
Open stope	18	64.2	170	1.17
Sublevel Caving	3	65.5	17	.95
Top-slicing..	3	59.8	37	.93
Caving....	4	60.7	7	.63

1 - These are average percentages from the Bureau of Mines Information Circulars on mining methods and costs. The total labor cost includes the cost of supervision chargeable to mining, which is from 2 to 3 per cent of the total mining cost.

2 - Figures taken from Information Circular 6503.

While improved technology is an aid to reduction of mining costs, the ability of the management to devise ways and means of encouraging labor to accomplish more per hour without reducing wages is of greater importance.

In the following pages there has been brought together material regarding labor for the manager of the somewhat isolated mine to think over. Although many of the statements are self-evident, it is hoped there will be found ideas that will be useful in making his organization a more efficient and happy one.

GENERAL STATEMENT ON ORGANIZATION

Ideal organizations where all employees are endeavoring to promote the company's interests are rare. This leaves much to be achieved by proper leadership. To attain leadership in the handling of employees a mine manager must be a student of human nature; must have tact in dealing with personnel problems, both with the staff and with the workmen; must take an interest in everyone, including the workmen's families; and in certain instances must be a sort of father confessee whose judgment and fairness is the gospel of the colony.

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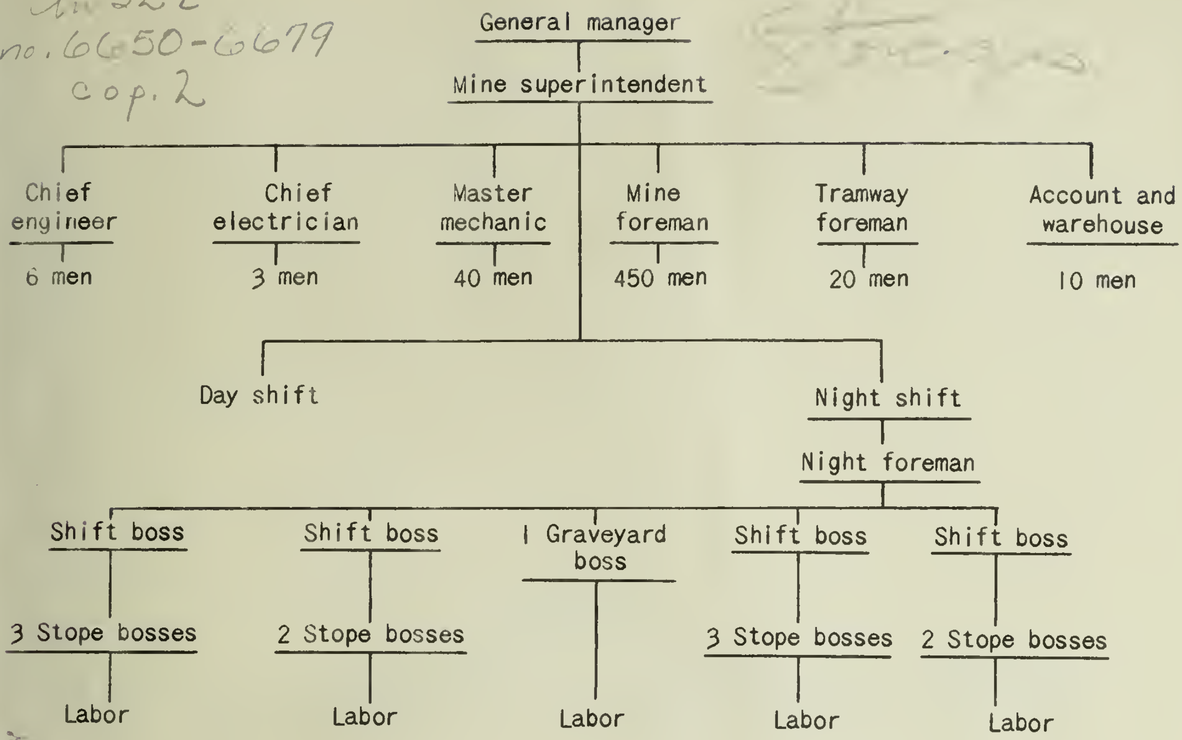

HARDY

Figure 1.- Organization chart of the Pecos mine of the American Metal Co. of New Mexico

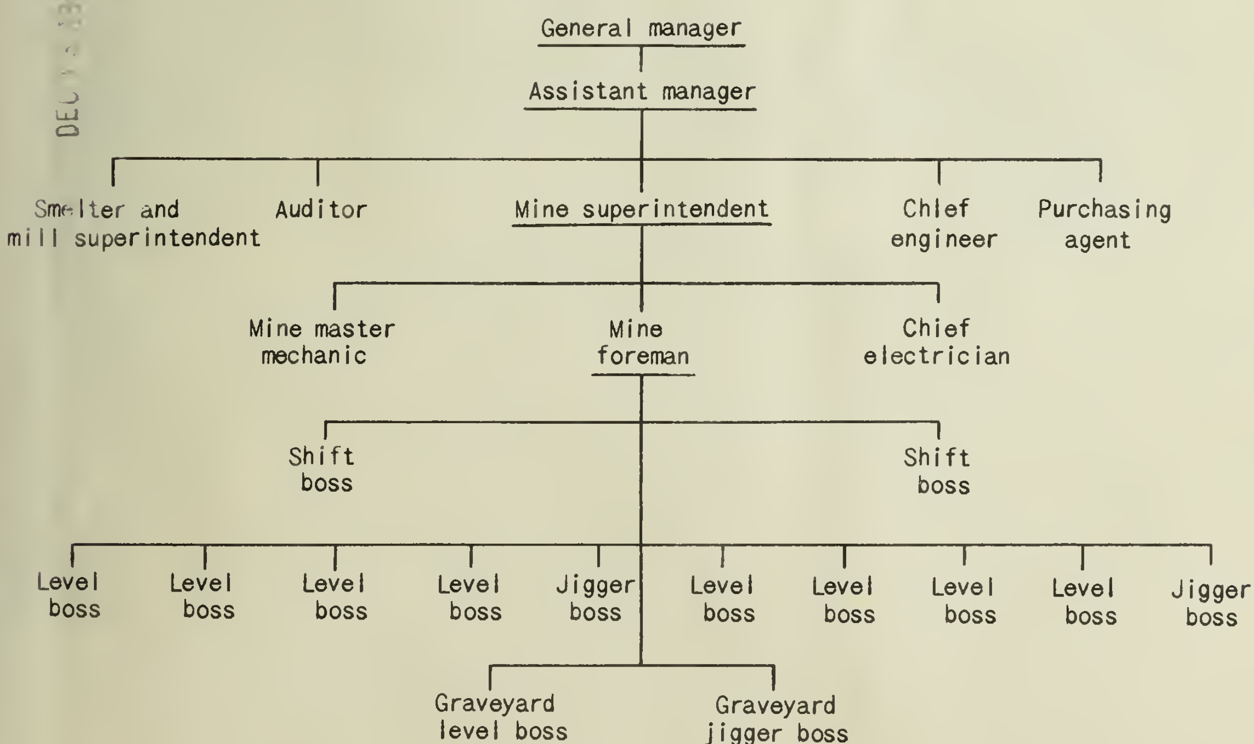

Figure 2.- Organization chart at the Magma Copper mine, Arizona

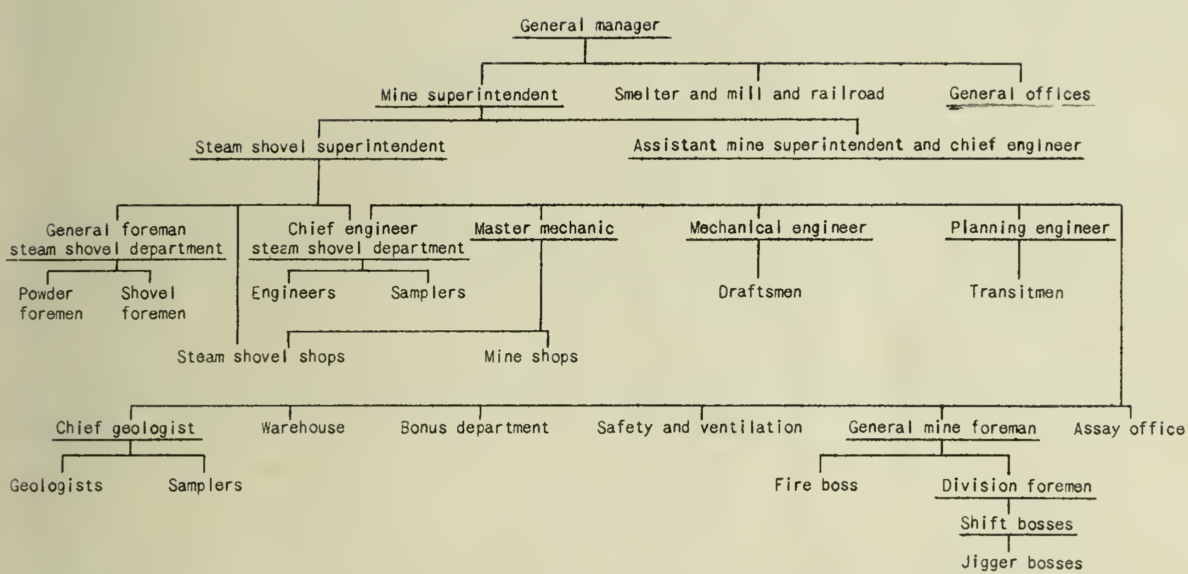

Figure 3.- Organization chart of the United Verde Copper Co., Arizona

The mine manager must have the ability to help his staff in solving their daily problems and to inspire them to use their best efforts and their ingenuity at all times in the accomplishment of their tasks, giving full credit to them for any constructive ideas they present which may be of value to the work.

One of the greatest problems in any organization is to forestall the development of friction and jealousy, not only among the workmen but between the members of the staff as well. This job of keeping peace among the employees is often the manager's most difficult problem and it is only through personal contacts with his employees that he is able to learn about the causes of such friction and to find a way to eliminate them. A few reliable men on the staff who are not afraid to tell the manager, himself, when they believe him to be in error as to decisions and who are ready to interpret the real feelings of the employees as to questions of the company's policy, will often save the manager and company from situations that might become serious. This does not mean that a manager should employ secret informers, which practice is not advisable, but he should recognize men of good will and capacity who have a sincere regard for the interests of the company.

ORGANIZATION CHARTS

To indicate how certain medium and large-sized mining companies distribute responsibility, organization charts are shown in Figures 1,[3] 2,[4] 3,[5] and 4[6].

There are about 500 men in the Magma mine, supervised as follows: 250 men under each shift boss, 30 to 40 men under each level boss, and 15 to 20 men under each jigger boss. The shift boss is directly in charge of the mine, and might be considered a swing foreman in other mines. The level boss is in charge of all work on a level or part of a level Jigger bosses are used on special jobs in comparatively isolated sections of the mine, such as where waste filling is being done, and also on haulage.

RESPONSIBILITY

The detail of control falls on the shoulders of those in active charge of the mine operations who are given full authority within their own spheres. They must answer to the general manager for the results of their work and must cooperate with the manager in correlating the work of their departments with that of the other departments. At the larger mines the mine superintendent appoints his assistants, foremen, and shift bosses, and requisitions for labor supply are sent through him to the mine employment office, but he and his staff should be permitted to act personally in respect to dismissals. At the smaller mines, the superintendent or foremen usually do the hiring personally. Orders should be issued through the officials immediately in charge of the workmen, and the responsibility of such official should be clearly recognized. The foremen and shift bosses must be reasonably assured that their superiors will support their actions and that their authority will not be questioned or decisions reversed in the presence of the workmen.

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3 – Matson, J. T., and Hoag, C., Mining Practice at the Pecos Mine of the American Metal Co., of New Mexico: Inf. Circ. 6368, Bureau of Mines, 1930, 21 pp. Fig. 18.

4 – Snow, F. W., Mining Methods and Costs at the Magma Mine, Superior, Ariz.: Inf. Cir. 6168, Bureau of Mines, 1929, 32 pp.

5 – Quayle, T. W., Mining Methods and Practices at the United Verde Copper Mines, Jerome, Ariz.: Inf. Circ. 6440. Bureau of Mines, 1931, 31 pp.

6 – Thomas, R. W., Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz.: Inf. Cir. 6167, Bureau of Mines, 1929, 27 pp.

Experience shows that years are required to build up a stable and effective force of engineers and workmen, so that there is a great need for continuity of employment. An experienced foreman can be discharged overnight, but to replace an efficient foreman requires months. It is therefore advisable to maintain a well-trained small staff which is perhaps less than could be used in prosperous times and more than is essential during periods of industrial stress, but which approximates that necessary for average operation over long periods. In this way, the men feel secure in their jobs, and are not hampered in their efficiency and creative effort by the ominous thought of unemployment.

SUBDIVISIONS OF SUPERVISION

For the purpose of supervision certain mines are divided into one or more subdivisions with a foreman in charge, and each subdivision has several sections, each supervised by a shift boss who is usually in direct charge of from 20 to 50 workmen. These workmen are divided into groups for specific operations such as drilling, loading, and tramming.

Another system used in some of the larger mines is to detail the shift bosses to supervise a specific underground operation such as development, stoping, drawing of ore from chutes, operations on the grizzly levels, or haulage. It has been found that this system is more effective, especially in those mines where the main operations are paid for on a contract basis.

CONFERENCES AND PREPLANNING OF WORK

The efficiency of labor at a mine depends in large measure upon detailed preplanning for the operations which each group of workmen is to perform, and in the pulling together of each group, or the coordination of their efforts, in speeding up the work. To do this effectively each group of workmen must be given detailed instructions by the shift boss as to the work they are to do each day and as to how they are to cooperate with the other groups.

Many workmen have good thinking caps and they should be encouraged to use them. The wise shift boss will get the men under him to express their ideas as to the ways and means of doing the work and will pass on to his superiors any suggestions that will improve methods of operation, at the same time giving credit to the workman offering the suggestion. The engineers depend somewhat on the foremen for such suggestions and ideas about the work and they in turn pass them on to the mine superintendent.

At some mines the manager or mine superintendent has a daily or weekly conference with the engineers, foremen, and shift bosses to discuss past results and mistakes that have been made, also to get suggestions that might improve the results. If these men, responsible for the work in the different sections of the mine and surface plant, show a generous spirit of cooperation and bring with them constructive ideas for discussion, these conferences are a real benefit, as they tend to unite the operating staff in concerted effort and make each individual feel more keenly his responsibility in the organization.

In order to assure the employees that their ideas will receive recognition and that they will get some credit for them, certain companies keep a box for suggestions, and the employee whose proposals are considered of value is given a week's extra pay or some other compensation, as well as public recognition.

THOSE WHO GIVE ORDERS AND THEIR ATTITUDE TOWARD THE WORKMEN

The men who give orders and are in direct contact with the workmen are the foremen and shift bosses, and it is their duty to see that orders are carried out. The most successful

boss is one who appreciates the difficulties and dangers that underground labor is subjected to and tries to help his men in every way possible. He does not drive when he should lead. Good bossing is a compromise between personal force and diplomacy. Workmen respect a boss who is a hard pusher if he is firm in his dealings and absolutely just. To win their loyalty the boss should take a personal interest in the men under him, know them by name, and see that they make good wages by showing them better ways to do their work.

THOSE WHO CARRY OUT ORDERS

Selection and Grouping of Personnel.— The success of any mine organization depends in a large degree on the efficiency of the workmen, who, as a class, have an innate tendency toward inefficiency and who, unless there is effective supervision or an incentive to work hard, will render as small an output as they can and still retain their jobs.

In selecting workmen it is nearly as important to do so on a basis of their fitness for each particular job as it is to engage a mine superintendent or engineer according to his merits. Some mining companies have an employment office through which requisitions for labor are filled, but the right to dismiss a workman is usually left to the mine superintendent or foreman. Certain mines give a special course of training to new men in the operation of rock drills or other machines and the shift bosses are instructed to spend some time where necessary in teaching the workmen how to do better work.

In many mines the workmen are grouped according to their task into squads of drillers, trammers, shovelers, timbermen, etc. These squads are detailed to certain stopes or mine levels and a close control is kept on the amount of work they accomplish. Charts are often kept to show these results and competition is thus developed between the squads and the shift bosses in charge of them.

It is the duty of the foreman or boss to determine whether a workman is worth his wage by comparing his work with the average performance of the other workmen. If there are poor men in a crew they should be supplanted as soon as possible by better men.

At some mines there is a periodic review by the management of all the employees, and this frequent survey of personnel as to ability and loyalty is found to be effective in maintaining a high degree of efficiency in the organization.

In cases where it is necessary for any reason to dismiss an employee, this should be done with a sympathetic understanding and a mind free from prejudice. Where necessary, helpful advice as well as reasonable money payments should be given if these are justified.

Machines, Tools, and Supplies.— As the continuity of operations depends upon the conditions of the machines in use, such as machine drills, scraper hoists, and haulage locomotives and cars, and on the availability of supplies, particularly drill steel and powder, pipe-repair outfits, timber, and tools, it is important that the shift boss or foreman does not overlook any of these details. He must see that machines, tools, and supplies are available when required. Good workmen soon become discouraged if they cannot have proper tools to work with and if their work is handicapped because of delays in getting tools and supplies due to lack of forethought by their superiors.

As a protection against theft or loss some companies sell the small working tools and lamps to the workmen at cost price and repurchase them if returned in good condition. The machine drills and other equipment are issued against a requisition signed by the worker's immediate boss.

Handling of Grievances and Discipline Cases.— At every mine complaints arise over questions of money earned, store deductions, working conditions in the mine, and living conditions. Often at the larger mines an officer is detailed to handle these complaints, and the mine bosses should give every aid to determine whether the complaint is just and to see that the workman is satisfied.

PAY SYSTEMS

General Use.- Among mine operators there are those who still believe that a straight day's wage with proper supervision is the best system, while others are convinced that wherever possible work should be done by contract.

A general survey of the pay systems in use is therefore of primary importance. The Bureau of Mines has issued a series of Information Circulars on mining methods and costs, and in most of them there is a brief description of how the pay systems are applied. There are about 80 of these papers, prepared by the mine superintendents or engineers, and from these circulars the mines have been grouped according to their pay system with the names of the principal mines where each system is being used. The result shows that -

16 per cent of the mines described were on a day's wage basis:

(Mines -- Utah Copper: Chino (New Mexico); Consolidated Cortez (Nevada); Kirkland Lake (Ontario); Tintic Standard (Utah); Central Eureka (California); Argonaut (California); Black Butte (Oregon); Nevada-Massachusetts (Nevada.)

8 per cent were on day's wage basis except mucking, which is done on a piece-work basis: (Mines in Tri-State district.)

18 per cent did development work on contract and all other work on day's wage basis:

(Mines -- Morning (Idaho); Engels (California); Pecos (New Mexico); Silver King (Utah); Page (Idaho); Groundhog (New Mexico); New Idria (California); Porcupine United, Vipond, and most gold mines in Ontario.)

18 per cent did development work on contract and other work partly on contract or bonus system and partly on day's wage basis:

(Mines -- Morenci (Arizona); Burra-Burra (Tennessee); Old Dominion (Arizona); Magma (Arizona); Cananea Consolidated (Mexico); Campbell (Arizona); Park Utah (Utah); Montreal (Michigan); Rosiclare (Illinois); Spring Hill (Montana); and others.)

40 per cent did practically all work on a contract or bonus system:

(Mines -- Mineville (New York); Alaska-Juneau (Alaska); Braden (Chile); Matahambre (Cuba); Marquette iron district (Michigan); Pilares (Mexico); Champion copper (Michigan); Edwards (New York); Mount Hope (New Jersey.)

Day's Pay.- When anyone pays a man by the hour or day, he buys human energy, but the workman does not always deliver the full amount paid for. The day-wage system encourages a man to withhold energy. He may work hard when the boss is present but will slack off and loaf when unobserved. This system encourages the boss to drive the man regardless of humanitarian, considerations.

It will be noted that the large open-cut mines, such as Utah Copper and Chino, are on a day-wage basis. At these mines the work is so organized and supervised that the men are required to accomplish just so much work within a specific time and a workman who can not keep up to the standard loses out. The ore cars must be loaded and ready for the trains on scheduled time. Adequate supervision is more easily accomplished in open-pit work than underground.

Piecework Systems.- Any contract or bonus system should function to the benefit of both employer and employee and should provide an incentive for the attainment of both quantity and quality of work and economy in the use of supplies. Carefully determined standards upon which a contract price or payment of bonus is based are essential for the success of these pay systems.

In nearly all of the systems, other than the leasing system, the workman is guaranteed his daily wage; the plan is to share profits, but not losses. He is not, however, guaranteed continuous employment should he fail to reach the standard of performance desired.

To make a man's earnings depend upon his proficiency elevates rather than lowers him in the industrial scale. To impose a tacit penalty for inefficiency dignifies those who are efficient and gradually eliminates the unfit. A continuous record of the efficiency of each man gives essential information to the manager, whereas a series of such records helps to indicate the degree of efficiency on the part of the management.

With each system, a variety of definite compensation scales is possible.

Contract System.- As employed in mines of the United States, the contract system is the method of payment whereby the miners are paid a set price per unit of work performed but are usually guaranteed the prevailing rate of wages should their earnings at the contract price fall below the wage rate. Contract prices are usually set at the beginning of each pay period, at so much per foot of drift, raise, crosscut, shaft or winze; per ton, per cubic foot, or per car of ore, or per foot of hole drilled; per set of timber placed, etc. The mining company usually furnishes all tools and all materials except possibly explosives. The contract prices are varied to suit conditions, taking into consideration the character of the ground, its "drillability" and "breakability" and requirements for support, and other factors influencing the difficulty of the work. In mining, these conditions are apt to vary widely in different parts of the mine and to change suddenly from point to point in the same section.

This system is aimed to reward the more able and energetic workman for his skill and added effort to promote efficiency, and if properly applied it usually produces these results.

Among the disadvantages of the contract system are:

(1) Increased earnings for the contractor may result from a sudden favorable change in conditions during the pay period, without any added effort or unusual skill on his part, and the company may then pay at the set price a greater amount than would be paid on a straight day's-pay basis.

(2) Contract prices may be changed frequently as conditions change, but a reduction in price during the pay period is usually resented by the worker. Likewise, when a reduction in price is made at the end of a pay period, during which excessve earnings were made due to favorable conditions, the contractor often feels that these earnings were largely due to his skill and increased effort, and that under the new reduced rate his efficiency is being discriminated against.

(3) It is difficult tc set prices for different classes of work and for different working conditions which will fairly represent the difference in skill and energy required for the performance of each class of work. To reflect adequately the various conditions in the prices, frequently requires somewhat complicated computations which the average miner is not able to understand, in which case he will probably be suspicious that he is not receiving the remuneration he deserves.

(4) The company does not participate in money savings resulting from speed, except in reduced unit overhead charges.

(5) Supervision is difficult, as the contractor is inclined to do work of poor and unsafe quality in the attempt to make speed.

In spite of these disadvantages, the contract system usually results in more efficient work than does the day's-pay system, especially for certain tasks which can be more or less standardized, where conditions do not vary widely, and where the management is able to pick and train certain men to do specific kinds of work. Its use means that the company can figure on a definite labor cost for specific pieces of work (except when the contractor falls below standard), and can expect a maximum of speed in getting the work done.

Leasing System.- The leasing system as practiced at Cripple Creek and other western camps is in effect a true contract system. Under this system, the miner is given a certain block of ground to work, the proceeds from the sale of his ore being divided between the company and the lessee on a predetermined basis. The lessee furnishes all the labor, usually the

explosives, and sometimes part of the tools. He receives no more than his share of the proceeds whether he makes wages or not.

Bonus System.- Under the individual bonus system, the miner is paid day's wages for the number of shifts worked. In addition he receives a bonus of so much per foot of heading driven, per ton or car of ore, etc., in excess of a prescribed standard performance which is usually taken as the average performance of a good workman under average conditions. The bonus is figured so that both the miner and the company benefit directly by units of work performed in excess of the standard task. The miners receive additional pay in the form of a lump sum, an additional number of shifts, or so much per shift extra, while the company gains additional units of work at less than the average cost and also a reduction in overhead (which also results from superior performance under the contract system).

The group-bonus system, based on group output, is one that can be readily introduced without friction, but there is not the incentive for each man to do his individual best, no reward for the better worker. The plan, however, does discourage any slackness on the part of the poor workers, as good workers will insist that all men do their share of the work.

The bonus system is frequently applied to gang work, where the men (and often their foreman as well) receive a bonus conditional upon the completion of the prescribed task within the standard time. The contract system, found in large mines, frequently involves either a gang piece rate or a gang bonus. A contract rate may be accompanied by piece work or bonus rates for the men working under the contractor; or the men may be straight day workers, having no share in the profits which the contractor derives from their labor.

Methods of Determining Contract Price or Bonus.- Each mine superintendent has to work out a basis on which he can determine the price to pay for development work, stoping, loading, tramming or any operation to be let at contract or performed on a bonus basis. The best basis is to know average past results and then to pick the best men for the work and arrange a contract with them or to institute competitive bidding among the mine contractors. In some instances time studies are made as an aid in determining the price to be paid for the various operations, but they are often misleading and good workmen object to having a man with a stop watch keeping time on their work.

Certain of the large mines employ contract or bonus engineers who decide the amount to be paid for a specific amount of work, while others have an efficiency engineer who studies the ways and means for improvement and advises the foremen in regard to these and also in reference to contract prices or bonus payments.

GENERAL PRACTICE IN PAYING LABOR IN THE PRINCIPAL MINING DISTRICTS

Mineville, New York.- The pay system in use at the Mineville magnetite mines is described by A. M. Cummings, mine manager, as follows:[7]

> At the Witherbee, Sherman & Co.'s mines the wage is based on an hourly rate.
>
> For 1927 the schedule as applied to the important classifications was as follows:

7 - Cummings, A. M., Methods and Cost of Mining Magnetite in the Mineville District, New York: Inf. Cir 6092, Bureau of Mines, 1928 pp. 10-11.

	Per Hour
Trammers	$0.32
Drill runners	.36
Hoistmen and locomotive operators	.34
Carpenters and repairmen	.40
Foreman and roofman	.41
Shift foreman	.45

A premium is paid all classes of labor. This premium is based on the hand cars loaded and trammed and the tons broken per man drilling.

Tramming.- Twenty-eight cents per car is paid to the men who load and tram 10 cars per shift, 29 cents per car for 11 cars and 30 cents per car for 12 cars and over. Men tramming less than 10 cars per shift receive the base rate of 32 cents per hour or $2.56 per shift.

The tram cars have a capacity of 1½ tons of the richest ore when completely filled, but due to the different grades of ore, an average of 1 ton per car is obtained.

Drilling.- An average of 30 tons per man drilling for a period of one month for all men engaged in drilling is the basis of the drilling premium. Fourteen cents per ton for the average tons per man drilling is paid each man engaged in drilling. When the average of the tons per man drilling falls below 30 tons, the men receive the base rate of 36 cents per hour, or $2.88 per shift.

The difference between the hourly rate of $2.88 per shift for drill runners and the amount earned at 14 cents per ton is the premium. This premium is given to all other classifications, including the mine captains, with the exception of the trammers, in addition to their regular hourly rates.

As far as possible the system described is made to apply to all conditions, but there are exceptions in both tramming and drilling work which make adjustments necessary in order that compensation for the work may be more comparable to the rate for average conditions.

In some of the sinks and long headings, special arrangements are made as to the number of cars required for a day's work. To make their earnings per shift higher extra hours are given the drill runners when in difficult places requiring the most experienced men. Extra hours are also given to those who assist in blasting. The hoistman and electric locomotive operators receive extra hours when the hoisting extends beyond the regular 8-hour shift.

This is a true bonus system, and it has the added feature, rather unique among mining companies, of being based upon group performance rather than individual output (except for tramming). The daily output is posted at the shaft so that the workmen know the results of their work as well as the bonus they will receive each day.

Edwards, New York.- The systems of payment at the Edwards lead-zinc mine have been described briefly as follows: [8]

All underground work except maintenance and repairs; installing chutes, track, and pipe; loading and hoisting in the shaft; some tramming; pumping and warehousing; and supervision is done on a contract basis with minimum wages guaranteed.

8 - Knaebel, John B., Mining Practice at the Edwards Mine of the St. Joseph Lead Co., St. Lawrence County, New York: Inf. Cir. 6586, Bureau of Mines, 1932, p. 21.

Development contracts are paid on a footage basis. In lateral work the contract includes every operation except laying track and pipes; in raises it covers breaking only, and the ore is drawn and trammed on a separate contract or on company account.

Stope contracts are all paid on a tonnage basis, computed from the cars and average load per car trammed each shift. Tonnage hoisted per shift is determined by weighing at the mill head, and a record of the cars hoisted provides the data for prorating the tonnage from each stope. A few stopes are measured by the engineers to determine the tonnage mined, in cases where the ore is mixed in ore passes with material from other sources. Shrinkage-stope contractors are paid for three times the tonnage drawn.

Tramming and motor-haulage contracts are paid on a flat tonnage basis. Allowance is sometimes made for long trams on development contracts.

Bosses receive no production bonuses.

Sanitary contracts, which involve removal of the portable toilets from each level for weekly cleaning on the surface, are paid a flat rate on the basis of 5 hours per job.

Chutes, track and pipe installations, repair work, and similar odd jobs are done by a regular repair crew on company time.

Ore hoisting is on company account.

Contractors buy powder, caps, and fuse; other supplies are furnished by the company.

Lake Superior Iron Mines.- Most of the eight mines described in information circulars employ a "contract" system (based usually on cars of ore, but sometimes on footage driven in development) for practically all underground work; one uses "contracts" per foot for development and a 10 per cent bonus over day's pay for extra work in stoping. Wages are guaranteed. Day's pay is used for some maintenance and repair work.

The following excerpts from papers on underground mining on the Mesabi range of Minnesota and the Marquette range of Michigan are fairly typical of practice in those districts. According to Hazelton:[9]

> Practically all the ore is mined by contract. Each contract rate is established by the superintendent and mining captain, and the men are advised of the rate at the beginning of the month. No change is made in the rate during the month unless it is obvious that it is too low. If it is too high the men benefit by it. At the beginning of the next month different rates may be established. The men have been convinced that they will get a square deal and are willing to work the last few days of the month just as hard as they do during the early part. The rate paid is set high enough to include the cost of powder and other miscellaneous supplies except timber, as these items are charged by the powderman to the contract in question. The foregoing plan is for mining places. Footage contracts are used for drifting and raising gangs. Only a few gangs of miners work on 'company account' and then only under some unusual condition making it impossible to set a fair contract rate. Contract miners are never paid less than the established company account rate.

9 - Hazelton, W. D., Underground Mining Practice and Costs at a Mesabi Range (Minnesota) Mine Using the Top-slicing System: Inf. Cir. 6325, Bureau of Mines, 1930, p. 9.

Graff[10] summarizes the usual Marquette method of paying labor in the following terms:

All miners work on a contract system, the unit price calculation being a 4-ton main-level tramcar. Every contract in the mine is provided with a scraper, and the contract price per car ranges from $1.10 to $1.80. A few gangs of timbermen are employed on the company-account system of pay. The miners pay for explosives, carbide lamps, and other small supplies. The company furnishes drill machines, drill steel, compressed air, electricity, loading equipment, and timber The miners do their own drilling, blasting, timbering, and mucking.

Michigan Copper Mines.- The contract system used at the Champion mine is described by Albert Mendelsohn,[11] as follows:

With the exception of repair and maintenance gangs all men in the mine work on contract. About 71 per cent of the men on the underground pay roll make money on their contracts; the average pay per contractor is about 50 cents per shift above the "company-account" base.

Shaft sinking is paid for at a set price per foot sunk and covers mining, mucking, and timbering labor. Up to a limited number of pounds per foot sunk, powder is free; all powder used above that limit is paid for at cost by the contractors.

Drifting is paid for at a set price per foot of advance, the price varying for each different width of drift and covering labor of mining, mucking, and disposing of rock. Up to a limited number of pounds per foot of advance, powder is free, the amount allowed varying with each different width of drift; any excess powder is paid for at cost by the contractors.

Because hand sorting is practiced in the mine the stoping contracts are somewhat more complicated than the development contracts. Barren rock must be kept out of the copper cars, and copper rock must not be discarded as waste. The rule given to pickers is simple: Large pieces of rock containing any copper whatsoever that can be seen must be shipped as copper rock; fine rock must be shipped as copper rock unless ordered left in the stope by the boss. Cars of copper rock are inspected by the bosses and by a poor-rock inspector. Discard in the stopes is inspected by the bosses and the captain.

In the stopes the contract system is balanced in such a way that the best pay will be made by those contractors who carefully sort their rock. The organization in a stope consists of a miner and one or two pickers on each shift. The miners' company-account rate is 45 cents a day above the pickers' company-account rate. These men mine, pick, and do all the propping necessary in their stope. Each participates in the bonus earned for the month in proportion to the number of shifts he has worked. Engineers measure monthly the volume of the excavation made in each stope to determine the total rock broken in each stope and the average width of the stope. Underground records show the number of cars of copper rock shipped from each stope. The time books show the number of shifts of all labor working in each stope. From these data the engineers calculate monthly the bonus to be paid to each party of contractors.

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10 - Graff, W. W., Mining Practices, Methods, and Costs at Mine No. 5 of the Marquette Range, Michigan: Inf. Cir. 6380, Bureau of Mines, 1930, p. 7.

11 - Mendelsohn, Albert, Mining Methods and Costs at the Champion Copper Mine, Painesdale, Mich.: Inf. Cir. 6515, 1931, pp. 11 and 12.

The use of three tabulated rate schedules facilitates the work of calculation. The rate schedules cover:

1. The bonus per shift to be paid for a given tonnage of rock broken in a stope of a given width; the wider the stope, the greater the tonnage broken per man per shift for a given bonus.

2. The bonus per shift for a given tonnage of copper rock produced, after sorting, from a stope showing a given ratio of total broken to total copper rock produced; the greater the amount of barren rock that must be handled in sorting, the greater the bonus pay per shift for a given tonnage of copper rock produced.

3. The allowance of powder per ton of rock broken for each width of stope.

Ducktown, Tennessee.- The following description by McNaughton[12] illustrates briefly the application of the contract system at the Ducktown copper mines:

General.- All development work, with the single exception of the long crosscut to the Eureka mine, is driven on contract. The blocking (blockholing) and loading are partly done on contract and partly on day's pay. Thirty-nine per cent of the total shifts under this classification in 1928 were paid on the contract basis. Thirty per cent of all stope drilling was done on contract. All other mine labor was paid according to standard daily wage rates for each classification of labor.

Development.- Drillers in standard 8 by 8 foot drifts receive a price of \$5.50 per foot with a 50-cent variation either way for very hard or very soft ground. The ground classification is determined at the end of each pay period by the mine engineer, who is himself an experienced drill runner. The company pays for mucking these headings in the manner described later under "loading."

The sublevel drifts are contracted at the following rates:

\$6 per foot for the first 50 feet from dump point.

\$6.50 per foot for distances of 50 to 150 feet from dump. Fifty cents is added to or taken from these prices for hard and soft ground, respectively. The prices named cover drilling and mucking labor and dynamite. The drillers receive 60 per cent and the muckers 40 per cent of the net earnings.

For raises driven by two men, which is the usual practice, the price per foot ranges from \$5.25 for the first 25 feet above the level to \$8.50 per foot for distances 200 feet or more from the level. Price changes are made every 25 feet. A 25-cent variation from these prices is made for hard and soft ground.

Stoping.- Drilling in stopes is contracted at a price of 7 to 10 cents per foot of hole drilled; the price is varied according to the hardness of the ground The price is fixed for each stope and is altered only occasionally as conditions change.

Loading.- Loading includes blockholing on grizzlies, loading cars from grizzlies, and hand-loading from drifts, stopes, and robbing. The men employed on this work are shifted from one job to another as occasion demands. Payment is made on a per-ton basis; the tonnage is figured from the car count.

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12 - McNaughton, C. G., Mining Methods of the Tennessee Copper Co., Ducktown, Tenn : Inf. Cir. 6149, Bureau of Mines, 1929, pp. 13-14.

Prices paid range from 4 to 30 cents per ton, depending upon the loading method, amount of blockholing required, and car service.

Labor for blocking and loading from grizzlies is paid for at the rate of 4 to 7 cents per ton. The 7-cent rate was paid at the first grizzlies installed, where storage was provided below the grizzly and the ore had to be pulled through a chute. On account of the blocky nature of the ore it tended to wedge in the chute, and almost as much labor was required to pull the chute as to block the ore through the grizzly.

Hand loading from development drifts is paid for at rates varying from 20 to 30 cents per ton, depending upon the material handled (waste or ore) and length of hand tram.

The actual price paid for loading is set for each job and place upon recommendation of the mine foreman and authorization by the superintendent.

Company rates and contractor's average earnings for principal classifications, 1928

Classification	Company rate per shift	Actual contract earnings per shift
1. Drillers:		
a. Development..........	$3.92	$7.17
b. Stoping..................	3.92	6.06
2. Trammers (loading)	3.60	4.97
3. Timbermen.................	3.72	-
4. Motormen..................	3.92	-
5. Skip conductors......	4.88	-
6. Hoist engineers......	4.88	-

Mascot, Tennessee.- At the Mascot zinc mines, large contracts are let in much the same manner as at Alaska-Juneau. Coy[13] states in part:

Wherever applicable contracting is in effect. Some miscellaneous work of such a nature that it can not very well be contracted for is carried on company time. However, bonuses apply to all company-time work, as will be explained later in this paper.

The mill-hole contractor agrees to break and deliver ore into cars at a stipulated price per ton. The company furnishes drills, drill steel, and power; the contractor furnishes labor and explosives. The chute pullers in turn contract with the mill-hole contractor to draw his ore into cars at a price per ton. Settlement is made with the mill-hole contractor each week; all labor, including his chute pulling, and explosive costs are deducted, and the balance earned on the contract is paid him. An itemized statement of his account is furnished him each week.

Contracts in open stopes cover merely breaking and apply only to tonnage made available to shoveling tracks. Open-stope contractors are required to do all blocking for shovelers and to assist them in keeping their working place in good condition.

13 - Coy, Harley A., Mining Methods and Costs, American Zinc Co. of Tennessee, Mascot, Tenn.: Inf. Circ. 6239, Bureau of Mines, 1930, pp. 7-8.

The contracts carry no guarantee as to earnings. Modifications of contracts are made when, in the judgment of the management, conditions beyond the contractor's control justify change. As a guarantee of the faithful performance of the contract, a holdback of 30 per cent is retained by the company until a definite amount has been set up. This holdback is payable to the contractor only upon termination of the contract.

Contractors are responsible for secure roof conditions in their working places. If it becomes necessary, through negligence on the part of the contractor, for the company to do this work, deductions are made from this holdback to cover such expense.

Although most contractors pay their drillmen and helpers company-scale wages they have the privilege of increasing the compensation received by such men. The company sets a limit to the increase. Any such increased pay becomes a part of the contractor's costs and is deducted on his settlement sheets; hence, he tends to make such increases sparingly. The system, however, allows the contractor to give recognization where it is deserved.

Development contracts are based upon footage of advance per week. In most instances a definite distance to be driven is specified in such contracts. Usually both men are drillmen. Settlements are made each week, and their earnings are proportioned to the number of shifts each has worked during the week.

Slushing contracts cover breaking, dragging, and chute pulling. Payments are made on a tonnage basis, as in other contracts. The contractor supplies all labor and explosives and takes care of all blocking, cable splicing, and hoist repairing.

Motormen and motor couplers participate, in proportion to their earnings, in a bonus based upon the total weekly tonnage handled. Skip tenders and helpers participate in a like bonus.

The day-shift mine foremen participate in a monthly bonus based upon the monthly production of 60 per cent zinc concentrates. As the night shift is principally a chute pulling and shoveling shift the night-shift foreman's bonus is based upon total tonnage hoisted.

Tri-State District.- Most operations at the lead-zinc mines of this district are on day's pay, except shoveling, which is paid for at a fixed rate per "can" loaded. Some pull drifts and some shafts are driven on "contract" per foot. Day's pay is determined by a sliding scale based on a price of $40 per ton for zinc concentrates.

Netzeband[14] makes the following comments on the pay system used at one mine in the district:

All labor, except mucking, is based on an 8-hour day. Muckers are paid on contract, 11 cents per can of 0.6 ton on the main level and 14 cents per can on the lower level. The bonus is paid to men employed on the lower level because the ventilation is poorer and they can not compete on an even basis with the men on the main level. The following wage scale was in effect during 1928 when zinc ore prices stayed at $40 or under for prime western ore.

14 - Netzeband, Wm. F., Method and Cost of Mining Zinc and Lead at Mine No. 2, Tri-State District, Picher, Okla.: Inf. Circ. 6121, Bureau of Mines, 1929, p. 9.

Machine runners..........	$4.25
Machine helpers..........	3.75
Trammers and drivers..	3.50
Hoistmen......	4.75
Powdermen................	4.50
Roof trimmers...............	4.00
Screenmen.	3.50

This wage scale is based on $40 zinc ore. If the price goes to $45 and stays there for one week, all wages are automatically raised 25 cents per shift, and muckers are raised ½ cent per can. In the same manner, for every $5 raise in the price of zinc ore above $45 the wages are raised at the same rate, but for every drop of $5 the wages are reduced. Wages are not reduced when the ore price drops below $40, for this is the base price for the wage scale. A week is always allowed between the wage changes to make sure that the market will not fluctuate above or below the critical price.

An average experienced mucker will load 40 or more cans in an 8-hour shift. There are several men at the mine who will average 100 cans per shift, but they are exceptional men and are usually given the best working places.

This may be regarded as fairly typical practice in the district. Contract rates for mucking vary considerably at different mines, however, due to variations in underground conditions.

Southeastern Missouri.- At the two lead mines described in information circulars, all work is on "contract" per foot or per ton (with hoveling on bonus for work exceeding 20 tons per shift at one of them).

Details of contract rates prevailing in 1929 are given by Poston[15] as follows:

All mining operations at No. 8 mine are conducted on day shift only. The standard day is from 7 a.m. to 3.30 p.m., with 30 minutes for lunch, which is eaten underground.

Base rates for various operations and for an 8-hour day are as follows:

Shift boss..	$5.95
Assistant shift boss......	5.35
Mechanical shovel operator.................	5.80
Company driller	5.05
Motorman...	5.05
Hoisting engineer..................................	5.05
Skip loader...	5.05
Hand shoveler (minimum of 18 tons)..	5.00
Railroader.............	5.00
Pumpman........	5.00
Switchman...	5.00
Pipe fitter...	5.00
General underground labor....................	5.00
General surface labor..........................	3.85

15 - Poston, Roy H., Method and Cost of Mining at No. 8 Mine, St. Louis Smelting & Refining Co., Southeast Missouri District: Inf. Circ. 6160, Bureau of Mines, 1929, pp. 15-16.

The rate of $5.80 for mechanical shovel operator applies only when conditions for mechanical loading are adverse. The operator usually receives 8 cents per ton, and if the earnings at this rate exceed $60 per week, half of this excess is turned back to the company. This refund feature does not apply to the hand shovelers, who retain all they may earn under the following schedule:

Contract hand shovelers receive $5.00 for loading 21 tons, 5.55 for loading 22 tons, and 0.28 for each additional ton above 22 tons.

Company hand shovelers receive $5.00 for loading 18 tons, and 4.50 for loading less than 18 tons, provided the total is approved by the shift boss.

The shift boss's approval simply indicates that conditions were so unfavorable that the required 18 tons could not be loaded.

All breaking is done under a contract system. Each tonnage contractor is guaranteed the company rate of $5.05 per shift. This wage is seldom paid,. however, for the contractors make more when paid on the following basis:

For stopes 8 feet or less in height, 14 cents per ton of rock broken.

For stopes from 8 to 20 feet in height, 13 cents per ton of rock broken.

For stopes more than 20 feet in height, 10½ cents per ton of rock broken.

The rates just quoted are for places where the rock is to be loaded by hand shovelers. When the rock is loaded by mechanical shovels the contractor receives 1 cent less per ton for all rock loaded.

The breaking and loading of all tonnage, exclusive of drift tonnage, is paid for according to actual underground scale weights and not by car units.

Drift contracts receive approximately $4.25 per foot of drift advance, from which is deducted a certain percentage of shovelers' labor necessary to remove the broken rock from the drift. Fifty per cent of the drift contractor's earnings in excess of $72 per week is refunded to the company. Fifty per cent of the tonnage contractor's earnings in excess of $60 per week is also refunded to the company.

Wages are paid weekly by check on each Wednesday for all time worked to 7 a.m. of the previous Sunday.

All employees must be ready for duty at 7 a.m., but the contract shovelers may leave the mine at any time after they have done the minimum amount of work required by the company. All other employees end their shift at 3.30 p.m.

Practice at another mine in this district is described by Jackson[16] in the following terms:

All men are guaranteed the basic daily wage ($5.05 per shift). Contracts are figured at the old unit prices, and the amount in excess of the old daily wage times the number of shifts work is added to the present basic wage rate times the number of shifts worked to calculate the total amount paid in each case. Eight hours constitutes a shift.

Drilling.- Drillers are paid a set price per foot of advance in development work and in stopes and stope headings for the number of tons broken, as measured by the number of 2½-ton cars loaded, recorded by car checkers employed for that purpose. Drillers furnish only their labor and powder, which is sold to them by the company at a fixed price somewhat lower than the actual cost price, while the

16 - Jackson. C. F., Methods of Mining Disseminated Lead Ore at a Mine in the Southeast Missouri District: Inf. Circ. 6170, Bureau of Mines, 1929, p. 19.

company furnishes drilling equipment, tools, and compressed air.

Prices for drifting, raising, and sinking vary with the character of the ground, size of excavation, and accessibility of the working place. The machinemen carry their own steel to and from the shaft, no 'nippers' being employed.

In stopes the unit price varies principally with the height of the stope, though character of ground, accessibility of the stope, etc., may be given consideration also. This price is 16 to 18 cents per ton in low ore and 12 to 14 cents in somewhat higher ore, to a minimum of 8 cents per ton in high stopes of "bluff mining."

Loading.- Prices for loading ore in stopes, whether by hand or mechanical loader, vary with the height of stope, tramming distance, and character of the work. Hand loaders tram their own cars and ordinarily use a mule for transferring cars between the loop and the muck pile. Ordinarily the mechanical shovel operators tram their own cars in the same manner, unless there is enough muck to warrant mule drivers being employed.

The usual task of "score," as it is termed for a hand mucker, is 20 tons, for loading which he receives only the regular daily wage. In excess of this he receives a bonus for each extra ton loaded. In places where there is a long tram hand tramming, or badly scattered muck, the score may be reduced to as little as 14 or 16 tons.

In development work it is customary to pay 8 to 10 or 12 hours for mucking out the round, depending upon the size of the excavation and loading conditions. Where the muck is all in one pile, as at the foot of a raise, and other conditions are normal only 8 hours are usually paid, while in driving a slope where loading is not so easy as in the drift, and where the muckers have to go up and down the slope to operate the hoist 12 hours would be the average pay.

Butte District, Montana.- A highly developed system of piecework, based on linear feet and cubic feet as the work requires, is used in most of the Butte copper mines. North Butte was one of the pioneers in developing standards for underground work. At Black Rock, the pay system used has been described by McGilvra and Healy[17] as follows:

> The wages paid at the Black Rock mine are the same as are paid throughout the camp. The wage scale fluctuates, and in the last 7 years the minimum daily wage paid underground was $4.75 and the maximum $6. The wage scale effective in the district in May, 1930, for the principal classifications of underground employees was as follows:

Hoist engineer (surface)	$6.50
Miner	5.25
Mucker	5.25
Motorman	5.25
Pumpman	6.00
Pipeman	5.75
Shaftman (sinking)	5.75 (contract work)
Shaftman (repair)	5.75
Station tender	5.75
Timberman	5.25
Trammer	5.25

17 - McGilvra, D. B., and Healy, A. J., Methods of Mining at the Black Rock Mine, Butte and Superior Mining Co., Butte District, Mont.: Inf. Circ. 6370, 1930, pp. 11-13.

Practically all breaking, shoveling, and timbering operations are done on "contract," as the term is used. It is really a bonus system and not contract work, as the men are given a certain definite price for the work to be done and if they make more money than the day's-pay rate they are paid the extra money as a bonus at the end of the period. If the men fall below the day's-pay rate they are guaranteed a minimum daily wage, but the particular place is investigated to see whether the failure to make bonus is due to conditions or price.

The samplers who take daily samples underground also keep a close check on all contract work and carry notebook sketches with them at all times showing a plan view of the working place. Notes are made from time to time of the progress of the work. The engineering department staff also aids in the measurement of contracts. Contracts are measured and calculated twice per month, but the bonus is paid once per month.

Prices are set by the chief contract engineer, and all contracts are checked by him at the middle and end of each month. A large sheet is kept in the office on which a plan of the work being done is sketched, and all calculations are shown so that the miners themselves can see what has been paid.

The price for drifts is generally $7 per linear foot, an over-all price that includes breaking, shoveling, and tramming. All timbering is paid for separately at $5 for each set of timber.

All timber for contract work is trucked by the miners at a price ranging from $0.50 to $1.50 per set, depending on the distance and conditions.

Nearly all raises are paid for at the rate of $0.080 to $0.100 per cubic foot of completed raise. This price is split up into $0.050 to $0.085 per cubic foot for breaking, $0.015 to $0.020 per cubic foot for timbering, and $0.015 to $0.020 per cubic foot for lining chutes.

Stope contracts are paid at a price ranging from $0.065 to $0.100 per cubic foot. The price varies, depending upon whether the rock is handled by sliding, mucking and sorting, silling, or underhand stoping. The over all price is divided into prices ranging from $0.025 to $0.040 for breaking, $0.025 to $0.040 for shoveling, and $0.015 to $0.020 for timbering.

Other operations, such as stope filling, $3 to $6 per set; bulkheading, $5 to $10 per set; raising chutes and manways, $12 to $16 for chutes and manways; and tramming, $0.015 to $0.030 per cubic foot, are also on contract.

Shaft retimbering and removal of old shaft timbers are paid for at the rate of $14 to $20 per linear foot for removing old timber and putting in new.

The wide variation in prices depends on the hardness of the ground, method of handling ore, and general conditions at the working places.

The plan of making contracts where the men buy their own powder and are paid whatever they make without the guarantee of a minimum wage has been tried.

The bonus system now being used has been successful, and it is the best plan which has been tried thus far. If the rates are equitable and fairly uniform for similar work, the miner who understands his business is paid a premium for accomplishment.

Coeur d'Alene District, Idaho.- Three of the Idaho lead mines described in the Information Circulars give data on pay systems. At one, development is on "contract" with 5 per cent withheld until work is completed, to insure good work. The balance of the work is on day's pay.

At the other two mines, the plan is similar, without any mention of 5 per cent retention. Day's pay rate is based on the price of lead.

According to Foreman:[18]

> The employees of the Hecla and Star mines are paid a base wage, plus a bonus which is dependent upon the price of lead for the preceding month.
>
> All stoping in the Hecla and Star mines is done on company account. Drifts in waste, crosscuts, and raises in ore or waste are generally contracted. No standard price is possible for contracts, due to changing conditions. A contract is given to the entire crew and earnings are divided in proportion to shifts worked. Contract calls for labor only, as explosives and other material are furnished by the company. Raises are also contracts, as previously shown.

At the Morning mine the pay system in use has been described by Wethered and Coady[19] as follows:

> All labor is based on an 8-hour day. Wages are uniform throughout the district and are based upon a sliding scale according to the price of lead. During 1928 when lead sold for 6 to 6½ cents a pound, wages were as follows:

	Per day
Miners	$5.00
Timbermen	5.50
Timbermen helpers	4.75
Muckers	4.50
Motormen	5.25
Motormen helpers	4.75
Shaftmen	6.00

> No contract or bonus system is employed in actual stoping operations. All work is paid for at day's pay according to the classification under which the work comes.
>
> Shaft-sinking, skip chutes, station-cutting, etc., are usually contracted for at so much a linear foot or cubic foot. The main development drifts on the levels are generally driven on day's pay plus a bonus for speed. Raises and crosscuts are contracted for at so much a linear foot.
>
> The mine superintendent has charge of all operations underground. His plans are carried out in detail by three foremen and a corps of shift bosses, none of whom shares in the contracts or participates in any bonuses.

Utah Lead-Silver Districts.- At the Park Utah mine the contract system is used for development work (including mucking, tramming, and timbering) and per set for stoping. Stope filling and miscellaneous work are on day's pay.

At Silver King, development is on "contract" per foot, and other work is on day's pay.

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18 - Foreman, Charles H., Mining Methods and Costs at the Hecla and Star Mines, Burke, Idaho: Inf. Cir. 6232, 1930, pp 17-18.

19 - Wethered, C. E., and Coady, Leo J., Mining Methods at the Morning Mine of the Federal Mining & Smelting Co., Mullan, Idaho: Inf. Cir. 6238, Bureau of Mines, 1930, pp. 9-10.

At the Park Utah mine, according to Hewitt:[20]

The labor for the mine is composed largely of men who live in the nearby farming communities. About 60 per cent of the underground employees are native Americans, about 25 per cent naturalized Americans, and the remainder are of various European nationalities. Labor is efficient and does not constitute a serious problem. The labor turnover is very small.

The following is the wage scale in the district in August, 1929, for the principal classifications of underground employees:

Cage tender	$5.75
Hoistman underground	5.75
Miners	5.25
Motorman	2.25
Muckers	4.75
Nipper	5.25
Pipeman	5.25
Powderman	5.25
Pumpman	5.75
Scraper operators	5.25
Shaftman	7.00
Timberman	5.25
Trackman	5.75

Square-set stopes are contracted for at $9 to $12 per square-set, depending upon whether 6 or 8 men, equally divided on two shifts of eight hours each are employed on the contract. Contractors are required to drill, blast, stand and block timber, place mining floor, rig up scraper, and scrape ore into the chutes. Tramming may or may not be included in the contract. Filling of stope with waste is done on company account. Tools and supplies which are furnished by the company are brought to within convenient distances of the working places by company-account labor.

Cut-and-fill stopes are contracted at 5 cents to 7½ cents per cubic foot. A contract crew consists of six to eight men and is divided into two shifts. The contractors drill, blast, scrape ore into chutes, distribute waste filling, and lay blasting floor. Tramming may be included in the contract price.

Stull raises 5 by 8 feet in section are driven on contract at $2.50 to $3.50 per foot. A crew consists of two men, one man on each shift. The company furnishes a helper to assist the contractor in getting stulls up the raise. Pipe is put in on company account.

Untimbered drifts are driven on contract at $4 to $10 per foot. The variation in price is due to whether or not speed is essential and the working-conditions. If timbering is necessary, the contract crew receives from $5 to $6 per set in addition to their contract rate per foot. The contract crew does all the work necessary to advance the heading, with the exception of putting in the pipe and the track.

20 - Hewitt, E. A., Mining Methods and Costs at the Park Utah Mine, Park City, Utah: Inf. Cir. 6290, Bureau of Mines, 1930, pp. 15-16.

At Tintic Standard contracts are seldom let, according to Wade,[21] who states:

> Contracts here are virtually bonus systems. So-called contracts consist of verbal agreements to guarantee wages and also to pay a stipulated price per foot for direct labor. These contracts are let only on drifts, raises, shafts, and winzes and are really negligible; contracts let in 1929 amounted to less than 1 per cent of the pay roll and in 1928 there were none.

Copper Mines (underground), Southwestern States.- Development is nearly always on "contract" per foot with wages guaranteed; stoping is on "contract" per cubic foot or on bonus for extra tonnage; tramming is on bonus for extra cars over base number or on day's pay; timbering is on day's pay or "contract" per set stood. At Bisbee, most work is on a bonus system. Probably 80 per cent or more of all work is on a contract or bonus system.

Contract and bonus systems have become highly developed at these mines; many of them are broadly similar to each other but embody differences in detail which have been evolved to meet local problems of administration or because of the individual views of the managements. The descriptions cited below are deemed worthy of inclusion in this paper because they not only present a picture of efficient systems, but serve as well to afford a comparison between systems which have been developed under broadly similar but locally divergent conditions.

United Verde.[22]

> A fair amount of work done safely by an average man in a given unit of time is the standard for any particular job. The efficiency of the job is calculated as follows:
>
> If a standard for a job is one unit per shift, and a man accomplished $1\frac{1}{2}$ units per shift, he is rated as being 150 per cent on that job. He receives as bonus one-half of his increase in efficiency over 100 per cent, or 25 per cent of his day's pay rate. The company benefits by the other half. Putting this into a formula, it may be expressed as follows:

$$T_2 = P|S; \quad E = T_2|T; \quad B = 1|2(E - 100)\ W$$

S = standard (unit of work per unit of time)
P = work accomplished
T = time required for doing P
T_2 = time allowed for doing P according to S
E = efficiency in per cent
W = wage rate per unit of time
B = bonus paid

> Obviously, the major problem is to determine correct standards of work. The setting of standards depends largely upon past performance and upon the judgment of the man setting the standard. The men chosen to set rates must have had considerable experience themselves in doing and in bossing the work they are rating. Such a man inspires the respect and confidence of the workers. The foremen are consulted whenever a new standard is to be set.

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21 - Wade, James W., Mining Methods and Costs at Tintic Standard Mine, Tintic District, Utah: Inf. Circ. 6360, Bureau of Mines, 1930, p. 14.

22 - Quayle, T. W., Mining Methods and Practices at the United Verde Copper Mine, Jerome, Ariz.: Inf. Circ. 6440, 1931, pp. 21-25.

Since 1924, a good deal of work in the mine has been paid for on a contract system. Practically all of the new timber work, drifting, crosscutting, and raising is done on contract. However, what we call a contract is not a contract in the true sense of the word, because the agreement is verbal and the man assumes no obligation. The company agrees to pay a certain price for a certain unit of work, but the man is guaranteed day's pay no matter how much he accomplishes. Also, the company retains the right of complete supervision over the work and the manner in which it is done. The contractor has nothing to say about who his coworkers or helpers will be, though, of course, an attempt is made to put men together who have equal ability and who will work together to advantage. Each man is responsible to the company only and they are paid separately by the company. Two examples of contracts follow:

A. Credit 10 chutes at $10 =	$100.00	Deduct 5 shifts at $5.23 =	$26.15
	49.55	5 shifts at 4.68 =	23.40
Due contract	50.45	Total deductions	$49.55

Due timberman in addition to day's pay: 53 per cent of $50.45 = $26.75
Due helper in addition to day's pay: 47 per cent of $50.45 = $23.70

Division of the profit is made according to the ratio of the day's pay of the two.

B. A drift is driven under contract at a price of $5 per foot. This price includes mining labor and explosives only, as the mucking is let separately. The miner drives the drift 100 feet in 20 shifts and uses 30 boxes of powder.

Credit 100 feet at $5	$500.00	Deduct 20 shifts at $4.95	$99.00
	339.00	30 boxes powder at $8	240.00
Due contract	$161.00	Total deductions	$339.00

Miner receives $161 plus day's pay for 20 shifts.

Two advantages of this type of contract over the bonus system in use here are that the contract is easily understood by the miner and he visualizes his units of work accomplished in dollars, which acts as an incentive to greater industry. Also, in the case of development work, it rewards the miner for saving explosives, while under the bonus system the tendency is to be extravagant and wasteful of explosives.

This type of contract has been very successful in lowering unit costs and in increasing the earnings of the men. It has been tried to a limited extent in mining in stopes and within the next year will be given a thorough tryout for all extraction operations.

A great deal of care must be taken to set a correct contract price in this method of payment because the company does not participate in the bonus, and if the price is set too high the unit cost to the company will be too high.

A great many of our contract prices have been arrived at from the bonus standards, in conjunction with the average bonus earned over a long period of time on a given class of work. For instance, the price of a drift would be set as follows:

The bonus standard for the drift might be 2 feet per shift. The average bonus for drift work was somewhat under 50 per cent, but good miners, under favorable conditions, often made 50 per cent or better. To make 50 per cent bonus in the above drift, the miner would have to average 4 feet advance per day, and this would cost the company $4.95 plus 50 per cent of $4.95, equalling $7.42, or $1.855 per foot for labor. Then, if $1.90 per foot were allowed for explosives, the contract price would be $3.75 per foot. If the miner broke 4 feet per shift and just used the allowed amount of powder, he would make just the same amount of money as if he worked under the bonus system. To make more money on the contract than on bonus, he would have to reduce the amount of powder used per foot or increase the amount of advance per shift. If he fails to make an advance which would have paid 50 per cent bonus under the bonus system, or if he uses an excessive amount of powder, he will not make as much as if he were on bonus instead of contract.

The company gets the work done at a certain fixed price whether the miner makes a good bonus or not, unless he shows a loss. In case of a loss by the contractor, the cost to the company is increased by the amount of the loss over the agreed price.

If the contractor is not doing reasonably well, an investigation is made of the conditions governing the operation, and the contract price is changed if unforeseen difficulties have arisen. If, however, the fault is found to be with the contractor's ability, he is either transferred to some work he is better fitted to accomplish or he is discharged.

The contract price is subject to change at any time if conditions over which the contractor has no control are changed. For instance, the formation in a drift may change from soft to hard rock, in which case a new price would be set starting at the contact. If an error, due to misjudgment, has been made in setting the contract price, the change in price is retroactive and starts when the work started if the original price was too low. If the price was set too high, the cut does not take effect until the first of the month following the time the mistake was discovered. Although this gives the men an advantage over the company, it tends to inspire their confidence in the system.

Most of the underground work and all of the mechanical and open-pit work is still under the bonus system. Various types of standards or allowances are set for the different operations according to our ideas of what seems to be the most equitable method of settlement.

Type of work	Unit of standard
Underground work:	
Mining in drifts and crosscuts	Feet per shift advance
Mining in stopes	Cubic feet per shift broken
Laying track, hanging pipe, etc.	Feet per shift installed
Hoisting ore	Skips per shift hoisted
Hauling ore	Cars per shift hauled
Mucking	Cars per shift filled
Open-pit work:	
Trucking ore	Tons hauled per shift
Electric shovel operation	Cubic yards dug per shift
Truck maintenance	Dollars allowance per truck shift

Type of work	Unit of standard
Mechanical work:	
Machine work	Given number of hours allowed per unit of work
Sharpening drill steel	Time allowance per rock-drill shift

The above are only a few examples of the many operations, but they are representative. The drill steel sharpening standard is not based on the number of pieces of steel sharpened, because when this method of payment was tried, the operators burned the steel in an effort to increase their production. When burned steel went underground, the bit soon failed and the steel was out again for resharpening. This tended to increase the work in the sharpening shop, and thus the men were paid a premium for doing poor work. Under the present system, when perfect steel is sent underground, a minimum amount of resharpening is required, the shop crew is as small as possible and a maximum of bonus is made by the men, so that the premium is now paid for good and efficient work. The same method of payment is used in the rock-drill repair shop. They are allowed so much time per drill in operation, so that drills sent down in perfect order will stay down the longest time possible and reduce the work in the shop to a minimum, thus allowing the men a maximum of bonus. This method of reward does very well for maintenance and repair work where certain equipment is in constant use and requires constant attention.

The organization of the bonus department consists of a chief bonus engineer, an assistant chief bonus engineer, an office clerk who posts the time of the workmen on the various job sheets, four bonus engineers who handle the underground work, and two bonus engineers who look after the outside and mechanical work.

The engineers take the cubic feet broken, explosives used, cars trammed, etc., from the shift bosses' distribution sheets each day. They are constantly visiting the working places, so that they are fully acquainted with the details of the work going on. Each engineer computes the efficiency of all the jobs under his jurisdiction, and the computations are then checked by the head of the department or the assistant.

The number of men required to carry on bonus work depends largely on the number of jobs. In turn, the number of jobs depends on the amount of attention which it is thought advisable to spend on the details of each major operation. For example, the shops may be given the job of completely overhauling a Mallet locomotive. A standard of 500 shifts might be set for the total job. This would be one job and would require very little work from the bonus engineer after the standard was set. The job might be divided into machine work, boilermaker work, steam fitting, and electrical work, making four jobs out of one, which would require practically four times as much work in the bonus department. Or, each individual operation on the overhaul job might be considered as a separate job, rates set, time kept, and the computations made on a great many details which would take considerable time on the part of the engineer. This last method of handling the bonus is the most accurate and is the method employed where the results obtained justify the cost of the clerical and engineering work. In some cases, it would cost as much to obtain an accurate standard on a particular job as it would cost to do the work. This is particularly true for work which is done once and is not repeated. In such a case no attempt is made to set a stand-

ard for the job. If the men doing such work have been regularly working on bonus, they are paid a flat rate of bonus for the time spent on the job. This does not furnish any incentive for faster work, but it prevents dissatisfaction on the part of the men required to do the work, and it is cheaper than spending the time to obtain a standard.

Some of the regular work about the plant is of such a nature that it is impractical to put it on a bonus basis, and such work is done on "company time," no bonus being paid over the daily wage rate. If a man works part of a shift on a bonus job and part of a shift on a company-time job, he will be inclined to charge most of his time to the company-time job so as to increase his bonus. The boss must therefore keep the men's time carefully and accurately.

The bonus is figured on the first of each month for the work done the previous month, and the money earned is paid on the check the men receive on the 21st of the month. In case a man quits during the month, his bonus is mailed immediately after the bonus pay day.

We have found that in every case where a fair standard has been set for bonus work, it has resulted in lowering the cost of that operation, and that the men doing the work earned more money than before.

Bisbee.[23]

At the Bisbee mines the bonus system is employed in a part of the development work and for some stoping.

In the case of development work, standards are set by the mine foreman and approved by the superintendent of mines. Bonus is paid on the measurements made by the engineering department. No penalty clauses for lack of making a standard are included, as daily wages are guaranteed.

In stoping, the bonus basis is set on tonnage per man-shift. This is determined from stope tonnage records.

Pilares.[24]

In 1929, 78.75 per cent of the workmen underground were on the contract basis. A contract wage scale has been prepared covering practically all classes of work underground, as well as many operations on the surface. All development work is on a contract rate per meter of advance. The size of the opening, class of ground, and working conditions control the variations in price. For raises the height of the raise is also a factor. In drifts and crosscuts the mucking contract is often separate from the drilling contract. The price is based on the meterage advance per round blasted, with variations for size of opening and distance for tramming.

All timbering in drifts and crosscuts is contracted at a price per set placed.

Stoping is contracted at a price per cubic meter. Different rates are paid for different types of stopes. There is a contract price for breaking, mucking, and timbering. In addition, there are prices covering all extra work, aside from

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23 - Lavender, H. M., Mining Methods at the Campbell Mine of the Calumet and Arizona Mining Co., Warren, Ariz.: Inf. Circ. 6289, Bureau of Mines, 1930, p. 12.

24 - Leland, Everard, Mining Methods and Costs at the Pilares Mine, Pilares de Nacozari, Sonora, Mexico: Inf. Circ. 6307, Bureau of Mines, 1930, pp. 28-29.

the regular operations in the stope, such as for the placing of bulkheads, umbrella stulls, extra lagging, and flooring, so that the contractor is paid for everything he does. This has resulted in a receptive attitude on the part of the contractor toward safety orders, suggestions requiring the placing of extra timber, or the performance of any other extra work in the interest of safety or better efficiency.

All contracts are given to one man. He is held responsible for the safe performance of the work, according to standard practices laid down, and hires the necessary men. This does not in any way relieve the boss in charge of the responsibility of seeing that the safest, best, and most efficient methods are followed. However, the bosses' dealings are confined to one man in each working place.

The company advances to all contractors underground the sum of 3.25 pesos per shift, which is charged to the account of the contractor. This amount is deducted from contractor's account on settlement day, which is every two weeks. Where powder is used the amount per unit required is allowed on the contract in money, and the contractor's account debited with the amount actually used. If the contractor uses powder judiciously a small profit will generally accrue to him.

Measurements of all development are made weekly by the engineering department, and a settlement is made the same week with the contractor. Stopes are measured by the engineers every two weeks, and settlements made the same week. As a protection to the contractors' helpers the contractors are required to furnish to the accounting department, prior to the liquidation date, a list of men employed and the rate of pay promised each. These amounts are passed to each man, and the contractors' accounts debited, provided there is sufficient money to cover. If not, rates are pro rata. Contractors failing to make sufficient money to cover the guaranteed daily wage of 3.25 pesos for three consecutive periods automatically lose their contract.

The contract wage scale is elaborate and covers practically every class of work of whatever nature, both underground and on surface, for which a contract price can be fixed. Variations in prices are provided to cover all ordinary variations in each class of work. Any unusual conditions involving special prices not provided for in the wage scale are referred to the mine superintendent for his approval.

This wage scale is the result of years of experience under the conditions peculiar to this mine, and it has been adjusted from time to time to meet changing conditions, with the idea of so arranging prices that equal effort on the part of the workmen in similar classes of work will insure equal compensation.

The checking of work performed is carefully tied down by engineers' measurements or count. In case of doubt the workman gets the break, and when prices are given they are not changed until the following measurement period, unless an erroneous price, unfair to the workman, has been made. In this case, it is rectified at once.

The result has been highly satisfactory, and it is seldom that a just complaint is ever registered by a workman.

Morenci.[25]

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25 - Mosier, McHenry, and Sherman, Gerald, Mining Practice at Morenci Branch, Phelps Dodge Corporation, Morenci, Ariz.: Inf. Circ 6107, Bureau of Mines, 1929, pp. 27-29.

For the year 1928, 72 per cent of all labor in the mining department was paid on bonus or contract systems. The other 28 per cent was on day's pay.

Contracts.- A standard set of contract prices is in effect throughout the mine. The foremen who let the contracts have copies of the contract schedules. These contract prices cover such work as drifting, raising, and winzing--both driving and timbering--laying track, digging ditches, undercutting and shrinking in stopes, and salvaging mining material. The rates are followed very closely and seldom changed. Unusual conditions are stated on the contract form and authority for changes from the standard must be obtained from the mine superintendent before a contract is let. All measurements are made on the 15th and 30th of each month, and the earnings are computed by the engineering department. All men on contract are guaranteed day's pay at the prevailing wage. The time and class of work done by each man on a contract is recorded by the shift boss.

The manner in which this contract system differs from systems employed at many mines lies in the way in which the proceeds of a contract are shared among the workmen participating. The earnings may then be divided by any one of four methods: (1) An equal division of earnings to all men concerned in the contract; (2) the payment of day's-pay rates to all according to their respective day's pay-rates, with an equal division of the surplus; (3) payment of day's-pay rates to all, then a division of the surplus in proportion to his day's-pay rate; (4) straight payment to each man for the work that he does, having separate prices on timbering, mining, mucking, etc. The choice of the method of dividing the proceeds of the contract is left to the discretion of the foreman, who is familiar with the work, its conditions, and the desires of the men.

Bonus.- A bonus system is applied to men working in stopes and on transportation crews. In arriving at the bonus rates past daily performances of men employed at various classes of work have been averaged to determine a "base" for each class. This base represents the amount of money that would be paid if a man does an average day's work, and corresponds to a day's pay. This figure divided by the number of tons representing an average day's work is called the "base cost per ton" (or other unit). If a man exceeds an average day's work he is paid one-half his "base cost per ton" for each unit by which he exceeds average day's work.

The base in each working place is subject to change at the beginning of a pay period if conditions should warrant it.

At the drill-sharpening shop the proposal was made to the men that one-half of any saving in labor effected by a reduction in the force would be divided equally among the men in the shop. This ruling had the effect of raising the quality of the bits turned out, because with a higher standard of bit there was less work in the shop and less men were required. The base number of men is subject to change at the discretion of the mine superintendent when any of the operating conditions undergo changes, such as variation in amount of work done in the mine, change in shop practice, change in shop equipment, or change in scale of operations. The drill-shop foreman was not included in this bonus.

The mine shift bosses do not receive a production bonus but do participate in an accident-prevention bonus based on a graduated scale for each 1,000 shifts supervised without an accident.

Inspiration.[26]

All development work can be contracted in small units to men who have the desire to increase their daily earnings by increased effort or the more intelligent application of their skill. Contracts in small units are desirable from the viewpoint of the company, but more particularly from the viewpoint of the employee. If a contract does not prove to be profitable, the man realizes that he does not have to continue the job for a long period. Although the contractor can not be paid below the standard wage paid in the district for the class of work being done, it has been found that when a contractor falls below the minimum rate of pay, and he knows this without being told, he is apt to fall below the standard in the quality and quantity of work done. That is where the company suffers if the contract is continued. With a small contract, adjustments of rates of pay can readily be made. The adjustments may be necessary on account of unforeseen ground conditions, or on account of supply and ventilation conditions which may have had to be changed for operating reasons. Each contract stands on its own basis. Usually it is true that the same class of work in a division is priced the same, but small units readily allow minor adjustments.

However, the great advantage of any contract system over any bonus system is that a contract system is based upon a unit of work done for a specified price. It can be terminated by either the employee or the company, since it involves an understanding by both parties and has to be mutually satisfactory. It should be remembered that the so-called contract system at Inspiration is just an opportunity for an employee to make more than the base rate of wage for the class of work being done. The contract should, of course, return a profit to the company as well as to the contractor.

To arrive at an equitable bonus system for mining operations is exceedingly difficult, and it is considered at Inspiration that it more readily lends itself to abuses and injustices than a well-managed contract system. After once started, a bonus system is harder to terminate or change than a contract system, as a contract is based upon a particular piece of work and the bonus upon a class of work.

Magma.[27]

During 1928 from 60 to 75 per cent of all work underground was done under contract or bonus systems. Rates per foot for drifts and crosscuts are based upon the size of opening, location, and general working conditions. All timbering in drifts and crosscuts is contracted at a price per set. Sliding scales for raises are based upon the height of the raise, the rate per foot increasing every 50 feet. Shaft sinking is done under a bonus system, the bonus being figures on $\frac{1}{4}$-foot advance. Stoping is contracted for at a price per cubic foot. Different rates are paid for different types of stopes.

All men on contract are guaranteed company wages. The man responsible for the contract in drifts, crosscuts, and shaft sinking is paid extra for managing

26 - Stoddard, Alfred C., Mining Practice and Methods at Inspiration Consolidated Copper Co., Inspiration, Ariz.: Inf. Circ. 6169, Bureau of Mines, 1929, p. 20.

27 - Snow, Fred W., Mining Methods and Costs at the Magma Mine, Superior, Ariz.: Inf. Circ. 6168, Bureau of Mines, 1929, pp. 25-26.

the contract. Each man draws the regular wage for the work he is doing; the remaining amount due on the contract is divided equally among those participating. Measurements are made by the engineering department on the 1st and 16th day of the month. The contract earnings are figured by the mine superintendent and the auditing department.

Contracts are let by the mine foreman subject to the approval of the mine superintendent and general office.

From Jan. 1, 1922, to October, 1928, the wage scale per shift at the Magma mine was as follows:

Miners..........	$4.95	Brakemen	$4.13
Timbermen.....	5.23	Pipemen..	4.68
Shovelers......	4.13	Cagers...	4.68
Hand trammers	4.13	Trackmen	4.95
Motormen.........	4.68	Nippers..	4.13

In October, 1928, a 10 per cent increase in rates was put into effect. At this mine, as in all the important copper-mining districts in Arizona, the standard scale or base rate is paid when copper is at or under 15 cents per pound. Informal increases of 5 or 10 per cent are made at each 2 cents per pound increase in the price of copper. Corresponding decreases are made when the price of copper is reduced.

Ray.[28]

Contracts.- A set of standard contract prices is in effect throughout the mine embracing all work that can be handled satisfactorily by the contract system. These prices cover drifting, raising, sinking, timbering, retimbering, undercutting and stoping, and the reclaiming of timber and track material. All prices are on a per-foot basis with the exception of stoping, timbering, and the reclaiming of timber and track material. In the case of stoping and undercutting the price is based on what is locally termed "vertical lineal feet;" that is, a base price is made for each vertical foot of stoping, the actual price for any individual stope being the vertical price per foot times the lineal length of the stope. In case of timbering, a price is given for each set. In reclaiming timber a price is paid on a basis of board feet of timber recovered and in the case of track, pipe, etc., the price is based on the actual linear feet recovered. The contract rates in effect are seldom altered and then only with the approval of the mine superintendent. Contract work is entirely in charge of the engineering department and is supervised by a contract engineer, whose duty it is to sign new contracts and to keep in touch with the work as it progresses during the month. Contracts are signed for a month's duration only. They are measured on the last day of each month, no excess earnings being paid to the contractors on the mid-month payday. All men on contract are guaranteed day's pay. The contractors themselves are directly responsible to the mine boss in those sections they may be working, and it is the mine bosses' duty to see that the work is carried on in a workmanlike manner. Such earnings as may be made in excess of the

28 - Thomas, Robert W., Mining Practice at Ray Mines, Nevada Consolidated Copper Co., Ray, Ariz.: Inf. Circ. 6167, Bureau of Mines, 1929, pp. 21-22.

daily wage are proportioned to the men working on the basis of their daily wage, except that the contractor himself is allowed 25 cents extra per shift for assuming the responsibility of the contract. All contracts cover labor only, and all supplies are furnished by the company. Contracts have been tried at various times to cover the drawing of muck from the chutes, all tappers in any one area being considered on the contract and the basis of pay being so much per car of ore produced. Such contracting improves the efficiency of the drawing operations, but is not satisfactory, as it is almost impossible to maintain control of the draw.

Bonus.- With the present contract system, which borders on the bonus system, very little actual bonus work is done. A bonus is paid trammers for the tramming of muck from chutes, and a definite amount per car is paid for all cars in excess of a certain number. The base number of cars depends on the length of tram. In addition to the contract price paid for drifting and raising work, a bonus is sometimes allowed on rush headings of so much per foot, providing a certain footage is completed during the month.

A bonus is also applied to work in which great difficulty is being experienced. This in effect raises the contract price, except that the bonus is not paid unless some agreed footage is completed during the period.

Open-pit Copper Mines, Southwestern States.- At the open-pit copper mines nearly all work is on a day's-pay basis. Some isolated jobs, such as construction, etc., are let out on contract at Bingham.

At the United Verde mine, however, bonuses are paid and are described by Alenius[29] as follows:

> An efficiency bonus based on the work performed is also paid for most classes of labor. Standards are set for each class of work and a record is kept of the performance of each man or group of men. The men are paid on a basis of one-half of the work performed in excess of the standard amount. For instance, if the efficiency is 150 per cent, the workman receives 25 per cent of his base rate as a bonus. In some cases the bonus is figured on the basis of individual work; in others the performance of a group is considered.

Southeastern Alaska Gold Mines.- The Alaska-Juneau is the o ly large mine operating in southeastern Alaska. Here 75 per cent of the underground labor is on contract and the other 25 per cent share in a bonus based on over-all tons per man-shift. The main haulage is on a sıbcontract.

P. R. Bradley,[30] consulting engineer for the Alaska-Juneau mine, states:

> It's the careful picking and training of men for a specific task, such as driving powder drifts, running up raises, etc., that has been largely responsible for the low costs at the Alaska-Juneau mine.

In his paper, Information Circular 6186, he states further;

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29 - Alenius, E. M. J., Methods and Costs of Stripping and Mining at the United Verde Open-Pit Mine, Jerome, Ariz.: Inf. Circ. 6248, 1930, p. 32.

30 - Bradley, P. R., Mining Methods and Costs, Alaska-Juneau Gold Mining Company, Juneau, Alaska: Inf. Circ. 6186, Bureau of Mines, 1929, pp. 10-11.

Bulldozing, loading, and tramming, and all mine developments as well as preparatory mining work, is done by contract. Some of the features that have made the contract system successful, are:

The guarantee of a reasonable base wage for contractor and men.

The close supervision of work and the prompt adjustment of price when justified by changed conditions.

The careful selection of men for contractors, and the prompt elimination of such contractors as are unable to earn more than base wages at a reasonable contract price.

The furnishing of good equipment and its maintenance in good condition, with plenty of steel and good air pressure.

Maintaining the same price for explosives, regardless of market change.

The inclusion of two or more working faces in the same contract whenever practicable to obtain more nearly average drilling and breaking conditions.

He then describes in detail the stipulations of the various contracts as follows:[31]

The contract system has proved highly satisfactory. The men make better wages and the unit costs are less than they would be under straight day's pay. Supervision is reduced to a minimum, and the inclusion of day's-pay employees into the contracts indirectly, through a share of the earnings, swings them into the contractors' stride. The general result is an "esprit de corps" and genuine interest in the work on the part of the underground crew.

<u>Ontario Gold Mines</u>.- Some mines are on an all day's-pay basis, most perform development on bonus or contract where possible, and a few are trying a bonus system throughout.

At the Teck-Hughes mine in the Kirkland Lake camp, according to Henry,[32] day's pay prevails underground. Henry states that:

All employees below the rank of foremen are paid by the day. The rates are:

Shift bosses....................	$7.00
Mucker bosses and chute blasters...	5.00
Samplers..........................	5.00
Drill runners, timberman, and scalers	4.75
Pipefitters, tracklayers..........	4.75
Shaftmen..........................	6.00
Muckers, trammers, and drill helpers.	4.25
Nippers, deckman..................	4.25

Contracts are rarely let and then only for shaft sinking.

Development is done on a bonus system. Shovelers are paid 37 cents per car and earn about $7 per day. Two drill runners are paid $3 per foot advance in drifts and crosscuts, and four runners are paid $6 per foot in inclined raises. Bonus earned is about $2 per shift.

31 - Bradley, P. R., Work cited.

32 - Henry, R. J., Mining Methods and Costs at the Teck-Hughes Gold Mines (Ltd.), Kirkland Lake, Ontario: Inf. Circ. 6322, Bureau of Mines, 1930, pp. 7-8.

Labor is about 40 per cent British, 40 per cent Finnish, and 20 per cent Hungarian. The bulk of the development is done by the Finns.

No labor union of any strength exists in the district, and no strike or labor trouble of any kind has occurred since 1919.

At the rates for bonus enumerated above, development is done more cheaply than by payment of only the regular rates of labor; the bonus provides an incentive for the newer employees who are only placed on bonus when they have demonstrated their ability to do good work.

In regard to the method of paying labor at the Vipond mine in the Porcupine district, Dye[33] comments as follows:

All work below ground except in drifts and crosscuts is done on day's pay. The wage rate is: Shift bosses, $7 per shift; miners, pumpmen, scalers, trackmen, cage tenders and powdermen, $4.80 per shift; timbermen, $4.80 and $5.05 per shift; muckers, helpers for machinemen, and helpers for timbermen, $4.24 per shift.

The driving of drifts and crosscuts is done on a bonus system. At one time the work was done on a so-called contract basis by which workmen were guaranteed the regular scale of wages but were given the work to do at a price per foot; they furnished the explosives and the labor for drilling, mucking, and tramming. Unless the price paid was frequently changed to meet the varying conditions encountered, it was found that circumstanoes over which the workman had no control either made it impossible for him to make any bonus at all, or the bonus worked out at a figure which was obviously more than a just reward for any special skill and effort which he might bring to the task. The company found itself in a position where it suffered all the extra cost due to any adverse conditions encountered and did not benefit by unusually favorable conditions, if and when these were encountered. The system now employed does not entirely eliminate this difficulty, but it does to some extent, and it does reward good work under all conditions and at the same time leaves the company in a position to share in the benefits when the work is being done in easy ground.

The bonus now paid is for footage advanced in any month in excess of 4 feet per shift worked, and is as follows:

The drill crew is paid $4 per foot for all footage in excess of an average of 4 feet per shift worked. An allowance of $2.50 per foot advanced is made for explosives and the crew is credited with one-half of any saving they make, or charged with one-half of any explosives used in excess of this amount. The bonus due the drill crew on the above basis is then split three-quarters to the drill runner and one quarter to the helper.

The muckers are paid 50 cents each for footage in excess of an average of 4 feet per shift.

<u>Chile Copper Mines</u>.- Most of the copper mines in Chile used a contract system for the greater percentage of their operations. A good example of efficient handling of labor is described in the paper on the Braden Copper mine as follows:[34]

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33 - Dye, Robert E., Mining Practice and Costs at the Vipond Mine, Timmins Ontario, Canada: Inf. Circ. 6525, Bureau of Mines, 1931, p. 8.

34 - Webb, J. S., and Skinner, T. W., Mining Methods and Costs at the Braden Copper Co.'s Mines, Sewell. Chile: Inf. Cir. 6565, Bureau of Mines, 1932, pp. 10-11.

In 1928, 70 per cent of all underground mine labor was paid on contract basis.

A standard set of contract prices is in effect throughout the mine, covering all such work as drifting, raising, winzing, stoping, undercutting, tramming, steel sharpening, and timbering, both for standing new sets and for repair work.

All men working on contract are guaranteed day's pay, earned at the base rate of the occupation involved.

A bonus is paid on rush work, such as driving drifts or tunnels, where speed is an important factor. This is arranged on a meterage basis, an additional payment being made for every meter driven over and above the predetermined monthly average for the particular type of ground in which the work is being done.

WELFARE WORK

A few examples of what some mining companies are doing in welfare work for their employees will show the general attitude of the mining industry toward labor.

In an article describing the welfare work of the Cleveland Cliffs Iron Co., by W. H. Moulton,[35] are the following statements:

> The well-being of the employee of any company is fully as important a factor in its successful operation as the condition of the mechanical or other equipment, and should receive just as careful consideration. Many plans are undertaken by The Cleveland-Cliffs Iron Co., in a spirit of friendship and good will toward its employees, as well as because of a rational understanding of their economic value.
>
> The first thing of importance is the condition under which the men work, both in the matter of health and safety. Efforts have been made to surround the men at the mines with healthful conditions, and everything possible has been done to promote the safety of the employees in the operation of the different properties.
>
> Equal in importance to proper working conditions is that of satisfactory living conditions. It has always been the policy of the company to provide good homes for the employees, at a very moderate rental.
>
> The matter of good sanitation has always been carefully locked after, both at the mines and at home locations. The excellent system of ventilation in operation at the mines is one illustration of the benefit which has resulted from a study of conditions. This improvement means better health and longer life for the men.
>
> An excellent medical service has been maintained to insure the best of care for the employees and of their families. Hospitals have been erected, served by the most faithful and competent physicians.
>
> Consideration has been given to those men who after long years of faithful service have at last fo nd it impossible to continue at work by reason of age or disability. Pensions have been provided for the men reaching the age of 65 and who have worked for the company 25 years or longer.
>
> A well-considered community life has a large share in the encouragement of ideals and every reasonable effort has been made to provide and maintain those conditions that will be the most helpful. Club houses have been erected with

35 - Industrial Relations Work of the Cleveland-Cliffs Iron Company. by W. H. Moulton. The Mining Congress Journal Oct., 1929, p. 771.

such facilities and arrangements as to serve the whole community. The club houses have the usual equipment of billiard and pool table, bowling alleys, game rooms, gymnasium and recreational halls, shower baths, and special rooms for the use of the women. One of the club houses in addition to the other features has a fine swimming pool.

B. C. Yates, general manager of the Homestake Mining Co., states as follows:[36]

> Social engineering is a comparatively recent branch of executive responsibility, but the Homestake company takes pride in the thoroughness of this phase of its operations. No expense has been spared to provide medical, recreational, and educational facilities, to prevent accident, to take care of the injured, to provide insurance against old-age survival, and to care for the dependents of employees. The company has also contributed largely to the support of church activities in the region in which it operates.
>
> Hospital and medical service is free to all employees and to their dependents. A pension system, established in 1917, maintained solely at the expense of the company, provides that an employee may be retired on account of old age, physical disability, or disease. Every effort has been made to encourage the development of an independent, free-thinking, and free-acting community of people, able to govern themselves as real Americans. As a result of this policy, the town of Lead has grown up to become a community of home owners who possess civil pride as well as loyalty to the company. The policy has ever been to recognize the rights of employees and to treat them fairly. Any employee, whatever his position, has ready access to the general manager to state grievance or to suggest a new idea in regard to his work. The regarding of loyalty and zeal has insured an unusual continuity of service.

STATEMENTS ON LABOR BY CERTAIN MINE MANAGERS AND ENGINEERS

According to P. R. Bradley, consulting engineer of the Alaska-Juneau mine, "It's the careful picking and training of men to do one job well that has resulted in the low costs at the Alaska Juneau mine."

Mr. Brennon, mine superintendent at the Britannia mine in British Columbia, where the output per man was doubled during 1931, states that the increased efficiency has been largely due to the contract and bonus systems and the necessity of the entire force, including the workmen, to speed up the work. All knew that if the production cost of copper could not be reduced to the market price, operations would have to cease. The workmen at Britannia are encouraged to use their heads, and their opinions are given consideration. This recognition makes them take a greater interest in their work. The management is naturally interested in keeping the more efficient workmen and is doing so by maintaining better quarters, a good trading store, movie house, men's club, and competitive sports.

E. T. Stannard, vice president, Kennecott Copper Co., in reference to the management of labor at the Braden mine, makes the following remarks:

> "Grief," as it is called in connection with troubles, is a sign of poor management. Choked ore passes, loss of time due to lack of cars, etc., can generally be traced to negligence on someone's part.

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36 - Executive Direction and Social Engineering, by B C Yates, Engineering and Mining Journal, Oct 12, 1931, p 291

> Workmen as a class tend to render as small an output of work and of as poor a quality as they can possibly get away with. However, we owe a certain responsibility to the men who have to do the physical work and should provide for their safety and study out methods to make their work easier. The boss most immediate to them should take a personal interest in his men, know them by name, and see that they make wages by showing them better ways to do their work. There must be concerted effort so that every man will do his part willingly and to the best of his ability. Such a system of approach will make the day's work easier for both men and bosses.

Scott Turner, director of the Bureau of Mines, in his annual review of mining progress, published in Mining and Metallurgy last month, states:

> As the cost of mining depends largely on the efficiency of the workmen, such questions as pay systems, handling of labor, control of operations, and mine-office management are being studied more than ever. It is an outstanding fact that those mining companies where the management has taken a special interest in the individual worker, supplying him with the necessary comforts of living and entertainment, and making him realize that he is a responsible part of the producing force, thus developing his loyalty, are the companies that have achieved the lowest mining costs, the lowest labor turnover and the lowest accident rates.

Mr. Lucien Eaton, from the Marquette iron district, states that "the secret of good management is to take away obstacles in front of men and make everything simple by careful planning beforehand; have a definite method to determine exact amount of work done; see that workmen on contract make a good wage, and that there is a confidence between captain and contractor for a fair deal."

In a paper presented at the February, 1932, meeting of the American Institute of Mining Engineers, the writer, who spent 20 years managing mines in Sardinia and northern Italy, stated:

> For efficiency in operation the management should first see that the best system is developed in respect to gaining a particular objective, and then plan ahead for the work in all of its details, instruct the men in the best ways to do their work, and give them the incentive to work.

The three essentials for the mine manager to follow are:

(1) Caution in selection of men and the pay system.

(2) Concentration in the planning and supervision of the work to see that nothing is lacking to promote progress.

(3) Consideration of the individual workman to see that he and his family are well taken care of.

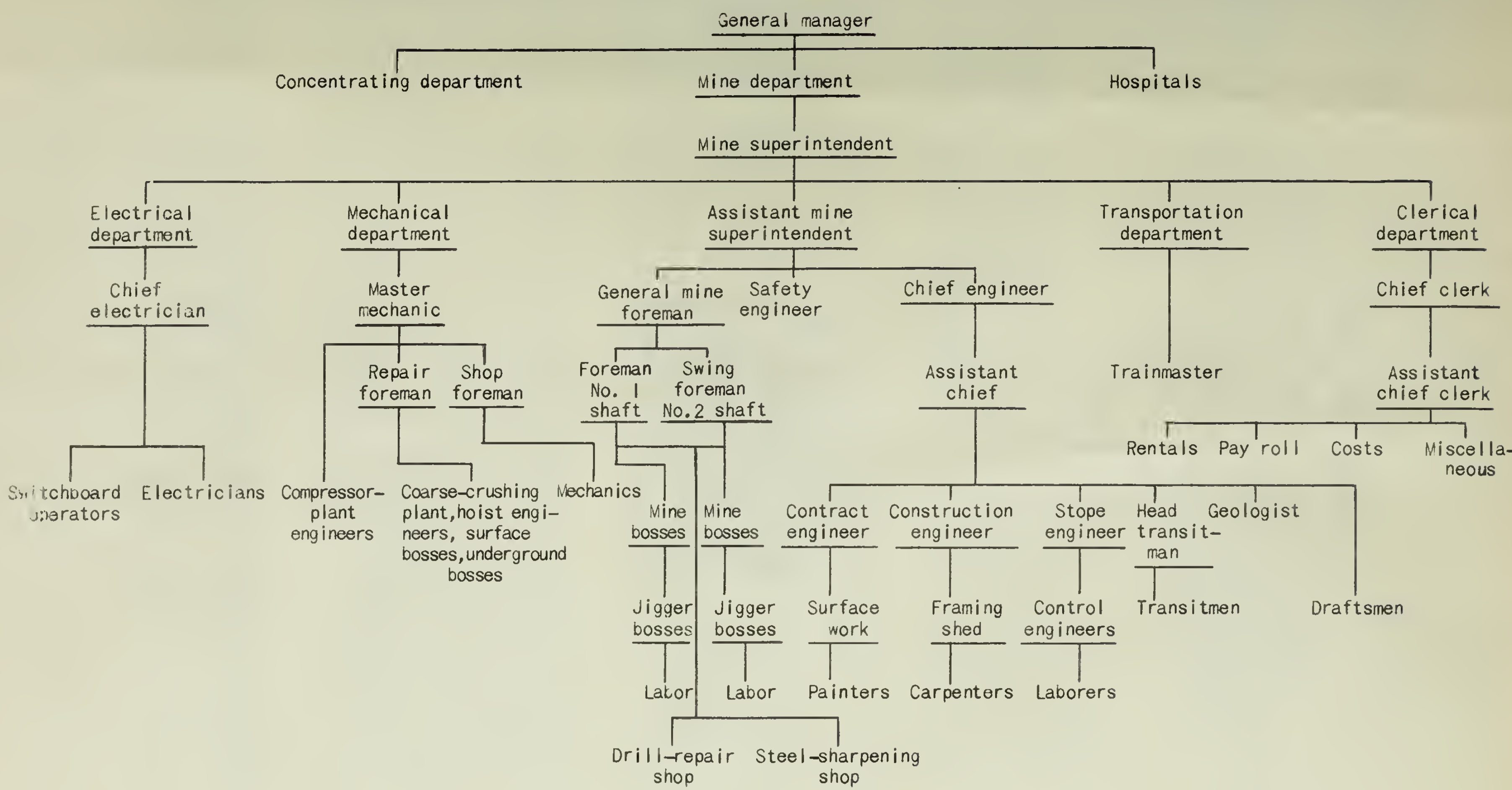

Figure 4.- Organization chart of the Ray mine of the Nevada Consolidated Copper Co., Arizona

25 oz.
40) $1000

50 oz
20) 1000

M. VAN SİCLEN
İKTİSAT VEKÂLETİ
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I.C. 6651
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DEPARTMENT OF COMMERCE - BUREAU OF MINES

ABSTRACTS OF RECENT ARTICLES ON MINE SUPPORT[1]

By W. R. Crane[2]

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RECENT PRACTICE IN SUPPORT IN MINES

Recent papers on support of roof and top rock in mines representing present practice in so far as it has been described by engineers in all of the principal mining countries, cover a relatively wide range of topics. The papers abstracted herein include such subjects as the testing of materials of support, failure and movement of rock above workings, effect of excessive pressures on rock masses, and the application in mines of timber, metal, concrete, packing, and filling support.

It is not within the scope of this paper to present a logical and orderly discussion of the elements of support and their application to conditions influenced by the material worked or the method of mining. Only the essential facts of a paper appear in the abstracts, the idea being to give a fairly clear picture of the behavior of various types of support under known conditions and to put into concise form important information concerning present practice, modifications of practice, and the trend toward new practice. With such valuable facts from others' experience made readily available, it is hoped that the engineer with particular and often trying problems may obtain suggestions that will be immediately helpful.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6651."

2 Senior mining engineer, U. S. Bureau of Mines, Pittsburgh Experiment Station,

GENERAL CONSIDERATIONS

Strength of Materials. Metal Ind., London, April 15, 1932, pp. 420-421. Tensile tests for brittle materials are of little value owing to the fact that there can be very little plastic readjustment, and it has been found very difficult to secure axial tension and uniform stress distribution for such materials. In the case of wood, it is so difficult to devise a test piece which will fail by tension rather than by shearing along the grain that the tensile test is rarely used.

The ultimate compressive strength of ductile materials is impossible to determine, but is sometimes reported by an arbitrary measurement of deformation. For brittle materials compressive strength is a fairly definite test result, although it probably represents destruction by shearing action, or by lateral strain, rather than by direct compression. It is probably a good index of service strength of brittle materials under compressive loads.

The cross-breaking strength can be obtained only for brittle materials and is correlated with tensile strength, but is frequently much higher than tensile strength tests, due probably to a slight flowage.

Further investigation of the physical properties of coal-measure rocks and experimental work on the development of fractures. Phillips, D. W., Trans. Inst. Min. Eng., London, vol. 82, pt. 5, pp. 432-450. The author discusses the behavior of rocks under varying conditions such as: (1) The effect of placing beams under a transverse load, (2) under compression tests, and (3) the production of experimental fractures.

The crushing strengths of coal-measure rocks are given and range from 4,000 to 20,000 pounds per square inch for sandstones and from 3,400 to 11,000 pounds per square inch for shales, which shows rather a remarkable variation in each.

The behavior of the roof, coal, and floor materials under forces operating in mines will depend on their relative strengths under these forces. The relative strengths vary in different mines and in different beds in the same mine. The tendency is to draw conclusions from the appearance of the immediately visible roof or top rock without considering what action may occur in the overlying rock mass.

No satisfactory results can be obtained by attempting to relate the mode of failure in small pieces of rock in a testing machine with what occurs in large masses of rock under stress in mines, such as, for instance, the formation of fractures due to advance of working face.

Results of reinforced column investigation at Lehigh University. Slater, Willis A., Eng. and Eng., October, 1931, pp. 219-228. In reinforced concrete columns the stresses in the reinforcement increase through flow of concrete. In general, columns that have a stress of 8,000 to 10,000 pounds per square inch at the beginning of loading may increase in stress to 42,000 pounds per square inch, with no indication of failure. Furthermore, tests show that

columns with plain ends -- that is, with uniform bearing on concrete and ends of reinforcement bars -- have greater strength than columns with other arrangements of reinforcing bars.

The extensive use of concrete columns in mines makes this information of particular value to miners.

Strata movements induced by longwall workings. Winstanley, A., Iron and Coal Trades Rev., London, February 5, 1932, pp. 246-247. Results of investigations with subsidence recorders, strain gages, and so forth, have made it possible to secure continuous records of movements of roof, floor, and in beds particularly in advance of longwall faces. It is shown that the movements are similar in character but different in magnitude in each instance.

As the working face advances, the roof beds lower under the force of gravity, but resist it to a certain extent and lower most over the area where the bed is being removed, transmitting part of the force over the solid bed, which decreases in thickness and yields outward at the free face; a similar action occurs in the bottom rocks. The significance of the converging movements taking place in advance of the working face is that the beds, when exposed at the face, have been subjected to distortion, and consequently have lost some of their strength properties. Roof beds when broken, in settlement, no longer offer resistance to gravity but become practically dead weights to be carried by the unbroken beds and the supports employed. Furthermore, the unbroken beds tend to distribute the effects of local forces, and thus avoid local damage to other beds. It is desirable that the roof beds be kept in contact one with the other in order to prevent extensive breaks close to the face; the closer they are kept to the waste pack the better. The earlier and more effective the reinforcement of the roof beds by artificial supports, the greater the chance there is of preserving the beds, thus requiring less support.

Ground movement in mines. McTrusty, J. W., Iron and Coal Trades Rev., London, March 11, 1932, pp. 435-437. Any practical policy claiming to have a scientific basis must be preceded by an investigation of the fundamental factors involved, the problems of safety and economy requiring an understanding of the cause of movement and logically improved methods of support and methods of working.

There are three areas involved in ground movement: (1) The subsidence area, (2) the middle active-pressure area, and (3) the outer or dead area. The types of movement do not vary much in the three areas, but change with the geological conditions, methods and rates of working, and the support adopted. The separate movements are interrelated and produce complex movement. The formation of faults has its counterpart in fractures produced by movements resulting from mining. All rocks are compressible at depth to an extent determined by their constitution; the energy of compression stored within the beds adds something in the nature of "live" loads to the dead weight of the material beds, hence the reactions of the strata liberated by

mining operations are not necessarily simple weight reactions. Time, pressure, and the conditions of restraint have to be taken into account when considering the flow or simulated flow of sedimentary rocks.

Fractures become inevitable whenever beds are strained beyond the resisting capacity of the material. Ordinarily the roof immediately above a bed at the working face is subjected to the weight of the superincumbent strata, and to compressive stresses from the cantilever action of the nether roof during subsidence; the presence or absence of natural fractures in the top rock must influence the concentration of stress.

Experience indicates that the safest roof conditions as well as the best working conditions are obtained where the roof over mined-out areas can be held for a considerable period close to its original position, which condition can be obtained to best advantage with systematic, straight-face working and adequate support. The success of any system of support is influenced, however, by the strength of the bottom rock on which it is erected.

The subsidence area in coal mines. Trans. Inst. Min. Eng., London, December, 1931, pp. 249-250. Subsidence is an inevitable consequence of extracting a seam of coal by the longwall method of working. The distance through which the roof subsides is roughly proportional to the thickness of the seam extracted, and, broadly, is about 50 per cent of the thickness when ordinary systems of packing are adopted.

Strong packs tend to reduce the rate at which the roof subsides, whereas weak packs permit an increased rate of subsidence. The force of the lowering roof mass within the subsidence area is practically irresistible, and ordinarily props are not suitable types of support for general use in roads within this area.

Surface damage through settlement due to mining. Booth, H. A., Coll. Guard., London, February 5, 1932, pp. 251-253. It is claimed by some that settlement of the surface following mining will cease in five years, but will depend upon the depth of the seam, the thickness of the seam, and the character of top rock. Furthermore, unmined pillars and props left in the workings delay the ultimate settlement. It is usually held that the surface may be affected to a maximum horizontal distance equal to one-third the depth of the seam; however, this may be influenced by proximity to old workings when the damaged area may be extended. The extent of damage is also affected by the character of the overlying beds and the dip of strata.

Rock bursts in the Mysore gold mines, India. Iron and Coal Trades Rev., London, April 8, 1932, p. 593. Rock bursts are of common occurrence in these mines and often occasion considerable damage to the workings, as levels and stopes, the former being more subject to such disturbances. Rock bursts occur without warning and with practically irresistible force, crushing large timbers and often demolishing several hundred feet of levels. The accompanying air blasts are serious and extensive, affecting workings for a mile or more.

The rock walls consist of a homogeneous gneiss, thousands of feet thick, which does not easily fracture and is not plastic. A theory advanced is that the walls are partially elastic and will yield to a very small extent on the removal of blocks of ground by mining, following which they remain stationary but continue to store up energy due to potential movement. Eventually the potential energy overcomes the resistance to fracture and the rock bursts.

No attempt has been made at scientific control of the roof in the mining of the lode, economic requirements being alone considered; and further, no support was employed in the upper levels, the effect of which on pressure in the lower levels is cumulative and adds greatly to the loads that are thrown upon such supports as are provided.

Rock burst in the Mysore gold mines. Min. Jour., April 9, 1932, p. 239. Much damage results from rock bursts in workings such as shafts, levels, stopes, and winzes. As a result of much investigation the workings have been made more secure and safe-guarded by the use of granite pack walls in stopes, and by the use of concrete and brick linings for levels and winzes. The value of such supports has been proved and their extension is steadily proceeding.

Pressure of rock masses in conglomerate lode. Vivian, Harry, Inf. Cir. 6526, Bureau of Mines, 1931, pp. 4 to 6. An interesting discussion of cause and effect of rock pressure in the deep workings of the conglomerate lode mines is given. The presence of joints, slips, and other breaks seem to deny the idea that unrelieved strains and stresses are the cause of violent breaking of pillars. Faults and slips contribute to the distribution and concentration of pressure in the workings, resulting in (1) a sudden increase of rock pressure upon certain pillars or temporary supports, (2) concentration of load at points on walls of fault, (3) shifting of loads on areas bordering caved ground to actively worked portions.

The weight of overlying broken rock acting on pillars within mine workings probably causes rock bursts, which will increase with depth, both in frequency and severity. The pressure block, consisting of broken rock above the stopes, determines the load upon the supports and consequently its amount and kind. The load on the temporary support is directly proportional to the time the support has been in place for a given position.

Roof support in bedded deposits. Walker, Arthur, Sci. and Art of Min., London, April 16, 1932, p. 308. Support should be given the top rock as soon as possible after the material mined is removed, in order to prevent separation of the roof beds. Such separation may in some instances extend over the bed mined for some distance. Temporary support is effective only in respect to the separated beds, while the beds above are capable of movement to the extent of the separation. The greater the extent of separation, the greater will be the impact of overlying beds should they be released through the extension of a break cutting the top rock. The separation of beds should be kept at a minimum in order to reduce the amount of such impact. The greatest rate of deflection is adjacent to the face, where the rate of separation is also greatest, for which reason the most effective supports are those set close to the face. Props set close to the face until space can be provided for more permanent supports check the deflection of the roof beds and materially aid support of the top rock.

Support of underground workings in Rand mines. Min. and Ind. Mag., South Africa, March 2, 1932, pp. 26-30. The dip of the roof varies from 5° to 20° on the far eastern portion and from 60° to 85° on the far western portion of the northern outcrop. The depth of mining operations is nearly 8,000 feet and support must be adequate for local needs, and also to maintain the integrity of adjoining mines. The types of supports employed comprise pillars, timber props, pigsties or cribs, mat packs or chocks, steel members, stone walls, concrete, masonry and brickwork, ore from shrinkage stoping, ore on stulls, waste from resuing, and sand filling. The method adopted depends upon the nature of the excavation, angle of dip, depth of workings, nature of the orebody (particularly the hanging wall), and thickness of the orebody. Props, pigsties, packs, and concrete, are used on dips of 0° to 35°; packs supported by pigsties or stulls, on dips of 35° to 45°; stulls against props for over 45°; stulls on skeleton pigsties or mat packs if the ground is heavy; the Randfontein method, which consists of spaced props, mat packs, and stulls for receiving the broken ore, placed at more or less regular intervals on dip and strike; and finally, shrinkage.

Increased rock pressure has made pillars unsatisfactory except as they may be employed in conjunction with other forms of support. Artificial supports have the advantage of being strong, tough, and durable. Timber is still the mainstay for underground support, the roughest kind of timber has been found adequate for props, pigsties, chocks, and various modifications. Steel has not been found desirable for support, while concrete, especially the concrete disk, is eminently suitable for certain dips and pressures. Stone packs commend themselves mainly on account of cheapness. The Sloan wire pack is a useful and often satisfactory type of support. Sand filling was formerly used mainly for strengthening pillars, but on account of its steadying effect it has been widely extended for ground support.

A method of preventing crush to brick stoppings and similar structures. Leeds, C. H., Trans. Inst. Min. Eng., vol. 82, pt. 4, pp. 310-311. Brick stopings or walls built in unsettled ground often fail because of roof pressure or creep, and the consequent effect upon maintenance of workings and derangement of ventilation may be disastrous. It is better to take precautionary measures than to rebuild supports.

It is claimed that stoppings may be built that will largely obviate failure and the expense of rebuilding. A foundation is prepared in the ordinary way and the brickwork built up to a height of 12 inches. On top of this wall, bricks are laid on edge flush with the face and back so as to form a trough 21 inches wide. The trough is then filled with peat moss, and upon which are placed undressed boards 27 inches long by $\frac{1}{2}$ inch thick by about 5 inches wide, forming a complete covering from end to end of the brickwork. Building of the brickwork is then resumed upon the top of the boards, the thickness of the stopping being now reduced to 18 inches so as to allow the bricks on edge supporting the ends of the boards to stand out clear of the brickwork above them. When brickwork of sufficient weight has been built over a portion of the length of the stopping, the moss is rammed beneath it in order to make

the combined structure as solid as possible, and the stopping is completed to the roof and wedged in with slate or other suitable material. As the weight increases, the boards break at the edges of the upper portion of the stopping and allow it to settle down gradually on the moss.

Effect of loads on props. Iron and Coal Trades Rev., London, February 5, 1932, p. 246. Measurements made on dynamometer props show that they are frequently swung under the influence of movements in the roof and floor. This causes eccentric loading, and the props are bent and damaged under loads much less than they could sustain when applied axially. The conclusion drawn is that props set in the usual way must not be expected to give full control to the strata movements. The variable and varying resistances of the props cause frequent local changes in the movements of the roof, and the props themselves are liable to become unstable; they nevertheless contribute appreciably to the control of the rock movements.

Effect of failure of caps and penetration of beds by props. Trans. Inst. Min. Eng., London, vol. 82, pt. 4, p. 334. When a cap splits or a prop penetrates a floor bed, its resistance decreases, and the material of the prop recovers under the reduced stress, which tends to the preservation of the props; but the loss of resistance is bad for the roof and floor. With unyielding end-contact surfaces, the shortening of a composite steel prop or even of a wooden prop is only a small fraction of an inch to the elastic limit of the prop. The yield to the convergence of roof and floor is therefore taken by the crushing of caps or penetration of the floor.

H-section props, with the same contact surfaces at the ends as cylindrical props, are weaker to resist bending at right angles to the web. A modified form of H-section prop with the flanges turned over to form an almost cylindrical section has been suggested.

TIMBER AS SUPPORT

Timber supports in the workings on the Rand, South Africa. Min. and Ind. Mag., South Africa, March 2, 1932, p. 27. Timber in various forms superceded pillars as the workings increased in depth, the eucalyptus, acacia, and cypress being the woods employed. The simplest form of support is the prop, which while temporary in nature has the advantage that it serves as an indicator of weight of the hanging wall. Props are readily adapted to varying dips and thicknesses of reef. Headboards are generally employed at the top of props to serve in cushioning and distributing the weight.

Pigsties are still considered the most efficient and economical support for dips up to 50° and for all depths. As usually constructed they are 4-sided and are left empty or filled with rock; skeleton pigsties are favored for dips of 45° and over. To insure strength and compactness the timbers are notched near the ends.

Mat packs or chocks are built up of short lengths of timber, (about 2 feet, 8 inches) and are wedged in place; they are usually employed in narrow reefs. The "duplex mat" is a square mat pack formed by superimposing layers of four lagging holes, spaced 4 inches apart and held together by rods.

Rough timber is ordinarily used and has been found adequate, but occasionally squared timber is found necessary, particularly where joints are required.

Use of props in coal mines, England. Coll. Guard., London, April 1, 1932, p. 635. The use of props in coal mines differs but little from similar use in all bedded deposits, consequently the following suggestions are of interest: (1) Rules relative to maximum distances in placing props should be adhered to for best results; (2) as the props next to the face are probably the most important, great care should be taken in placing and spacing them; (3) caps should be prepared at the surface and be of uniform size; (4) props at the face should be staggered in order to decrease the area of the roof unsupported; (5) faces should be carefully and regularly inspected and points marked by chalk where props are needed; (6) steel props should be used whenever possible as face supports, as they offer an even resistance to the roof, and the probability of induced breaks is reduced; (7) steel caps (flats) should be employed particularly at points needing special support; (8) all supports should be removed from the gob in order to insure even subsidence of the roof.

Homegrown pit props. Coll. Guard., London, February 19, 1932, p. 346. The Forestry Commission, in England, is rapidly increasing the output of timber suitable for pit props, and much larger quantities will become available in the near future. There is a possibility that the output of homegrown Scots pine, larch, and Douglas fir for pit props can be raised to 500,000 tons a year and even more as the State plantation develops. The importance of uniformity is pointed out and the suggestion is made that care should be taken to avoid marketing props with tapered ends. The props should be standardized as to size and shape.

Cantilever roof protection. Queensland Govt. Min. Jour., November 14, 1931, p. 437. As a means of protecting men working under loose or broken ground, a cantilever of timbers is used which consists of two 6 by 6 inch booms, held in place below the caps of the last two sets next the face of a drift by four strap-iron hangers. The booms can be slid back and forth in the hangers and with pieces of plank placed upon them crosswise form a protective lagging for the men. During drilling the booms are removed or drawn back in order to give plenty of room, and during mucking they are advanced to the drift face. Little space is occupied by the timbers and they can be put in place quickly.

Use of square-sets in the Butte copper mines. Harrer, C. M., Eng. and Min. Jour., March 1932, pp. 146-148. Square-sets and filling are commonly employed in the support of stopes in the Butte mines. The working face is carried up by offsets, and on the completion of an inclined slice, filling is introduced into the stope by merely knocking out or blasting out the sides of the end raise chutes facing the stope. The waste runs into the stope from the raise chute to its angle of repose.

The system of alternate stoping and filling has necessitated the division of the stope blocks between two raises into a double stope, one wing or side undergoing filling while the other is in the productive stage. Formerly, this method of mining was considered applicable only to steeply dipping orebodies and hard, firm, hanging wall; now stopes are worked with comparatively weak hanging walls by keeping up the filling and employing stulls.

Where the hanging wall is quite weak, a system of connected timbering, square-set, or stulls is employed as an auxiliary support. If the vein is rather wide, staggered square-set timbering is used. The inclined-cut-and-fill method has, therefore, a wide application at Butte.

Notes on timbering shafts in deep-seated placer deposits. Roberton, J. Hume, Canadian Min. Jour., October, 1931, p. 759. The size of timbers to be used in support of dirt, sand, and gravel deposits, usually saturated with water, is considered from the theoretical standpoint. It is estimated that for shafts 100 feet deep, timbers 14 inches wide by 15 inches deep would be necessary.

Use of timber for shaft lining. Eaton, Lucien, Eng. and Min. Jour., February, 1932, p. 85. The advantages of timber for shaft lining are (1) it is usually the cheapest and most convenient material to use, (2) it is easily framed and placed, (3) it is not eaten or corroded by acid water, (4) it gives warning before it fails, (5) when broken by pressure or accident, it can be chopped out and replaced with a minimum of difficulty or loss of time. Its chief disadvantages are (1) its bulk, for it takes up more room than steel, (2) the danger of fire, (3) its comparatively short life when it is not always wet.

Use of hanging bolts in shaft timbering. Eng. and Min. Jour., November 23, 1931, p. 452. The use of hanging bolts in supporting and holding cribbing in place in shaft sinking is described and sketches of them in use are given.

The preservative treatment of mine timbers. Vaughan, R. J., and Prettie, R. J., Canadian Min. Jour., October, 1931, p. 756. Wood has a wide range of use in mines, varying from temporary to permanent support, and an extensive application in secondary support -- that is, as an adjunct to all other means in maintaining mines. It is shown that failure of mine timbers in the mines of the United States is due to the following causes and is in the following proportions: (1) Failure due to decay and insects, 50 per cent; (2) failure due to breakage and fire, 20 per cent; (3) waste, 25 per cent; and (4) wear, 5 per cent.

Causes of decay are moisture and heat, especially under variable conditions of both; the most practicable means to be employed in preventing decay is by poisoning the food supply of the fungi -- that is, by the use of preservatives. An ideal preservative is one that is both toxic and volatile, the latter in order that it may be readily and efficaciously applied.

The paper deals with the use of zinc chloride and of creosote as timber preservatives, giving strengths used and method of application.

Bad air and timber decay. Min. Congress Jour., March, 1932, p. 26. It has been observed that when mines are closed for considerable periods, much expense is incurred in resumption of operations because of timber decay and the resultant caving of overlying rock. In certain instances it has been noticed that in the parts of mines in which there was good air circulation, there was a minimum of this decay of timber. The inference is that mechanical or controlled ventilation might pay if used only to protect timbers in both working and idle periods.

Portable timber saw. Coal Age, March, 1932, p. 128. A new type of the Wolf gas engine driven portable timber saw has been developed for cutting heavy timbers in the woods as well as for cutting and trimming mine timbers. These saws can now be obtained with any one of the three common power drives, as compressed air, electricity, and gasoline. Standard models have a capacity of 24 inches with a range of 18 to 48 inches. The gas engine used with the new type is of the aircraft design, being light and powerful.

FORMS OF METAL SUPPORT

Steel supports in the Rand mines, South Africa. Min. and Ind. Mag., South Africa, March 2, 1932, p. 28. Steel employed as support other than in shafts and as joists is not in favor in the Rand mines. When used as girders, or joists, the ends are inserted in hitches cut by machines in the solid rock of the walls of haulage ways. Often pigsties are built on top of the girders to support bad sections of the roof. The advantages of steel over timber are: (1) It is more durable and consequently cheaper to maintain; (2) it can be re-used several times; (3) is lighter and easier handled than timber of equal strength; (4) takes less room than timber; (5) does not decay and pollute the air; and (6) is noninflammable. It is, however, subject to corrosion from acid mine water and tends to weaken structurally under alternating and prolonged localized stresses.

Application of steel props in general. Coll. Eng., London, February, 1932, p. 69. As a result of experience in the use of steel props in English coal mines, their value has been proved particularly under favorable conditions of a clean seam 30 inches or more thick, with a variation in height of roof not exceeding 2 inches and having no dirt band with which to contend. With slight variation in height of roof a definite length of prop can be used. The initial cost of supports prohibits indiscriminate loss if economical working is to be attained. Furthermore, successful roof control demands complete withdrawal of all steel supports in the waste. It is therefore essential that a detailed record of props be kept.

Use of steel props in mines (England). Sci. and Art of Min., January 23, 1932, p. 210. The use of steel props is growing in the English coal mines, the forms in most favor being the H-section and the tubular types. The advantages of the use of steel props over wood are given as follows: (1) The height at the face is maintained owing to the rigidity of the props; (2) the roof breaks at the face are less because the props do not allow the roof to lower and fracture; (3) the stronger support that they affort ensures that the faces will be kept open; (4) they save the setting and withdrawal of wooden checks; (5) by being of uniform strength they prevent the formation of cross breaks; (6) better control of the roof is assured, giving improved working conditions; (7) there are fewer falls and a greater factor of safety; (8) less time is lost at the coal face, and consequently provide more regular work and better earnings for miners; (9) a more definite line of roof break is obtained; (10) minor coal face accidents are reduced, thereby obviating loss of time by miners.

In one instance where 6,500 steel props were used there was a loss of only 2 per cent per month, indicating that four years is the life of a prop.

Use of steel props in English coal mines. Luty, B. E. V., Iron and Coal Trades Rev., January 22, 1932, p. 137. Steel props are finding increasing favor, particularly the solid type and tubular form, the latter being considerably lighter than the girder prop of the same strength. It is also a fact that a considerable amount of squeeze can be taken without distortion of the tube; and when the wood ends are too badly mopped up for further use, a portion can be bored out and new end pieces inserted butting on the core so that the prop is again ready for use. Wooden cores should always be provided when tubular props are employed; while they may reduce the possibility of distortion, their real function is to simplify the reconditioning of the tubes.

Use of steel props in coal mines. Iron and Coal Trades Rev., January 29, 1932, p. 210. Owing to the extended use of steel props in English coal mines, it is interesting to note certain conditions under which they are not applicable. When the roof is weak and it is desirable that the weight should act slowly, the steel props resist it, with the result that they are pushed up through the roof and have to be abandoned.

Steel supports for roadways and steel props for face work. Trans. Inst. Min. Eng., London, October, 1931, p. 9. Failure of arch girders in roads is due to the following causes: (1) Improper packing of the crown of the arch; (2) side pressure forcing the crown upward and breaking the fishplates; (3) bad design of parts such as fishplates or too light arches; (4) improper spacing of arches; (5) imporper bracing of arches.

Lack of success in the use of steel props is probably due to use with other props rather than inherent defects in the props, themselves. Do not mix steel with wooden props.

Steel props and arches for roof support. Clements, Fred, Iron and Coal Trades Rev., London, January 1, 1932, p. 5. Owing to the nature of the stresses imposed upon props and arches used for roof support, it is imperative that the steel used should be (1) absolutely reliable, uniform and free from surface and internal defects, and (2) possessed of a high capacity for undergoing bending without fracture. Steel supports are subject to bending, which often ultimately and inevitably exceeds the yielding point of steel. Other things being equal, the capacity for undergoing bending without fracture is at a maximum in low-carbon steels, and decreases as the carbon content rises; therefore open-hearth steel is definitely superior to other kinds.

The use of camber girders. Trans. Inst. of Min. Eng., London, December, 1931, pp. 259-260. A camber girder consists of a suitable length of girder, curved in its middle portion but having a part of each end straight and tangential to the curved part. The camber is used in lengths of 5 to 12 feet and varies from 12 to 30 inches in the height of camber. The greatest height is generally used in openings made in weak top rock, but as a result of experience there is a tendency to reduce the height in proportion to the length of the girder. For instance, girders 11 feet long with a camber of 12 inches have proved successful in hard rock.

When the wall rock is reasonably strong, camber girders are secured in position by wedging the ends to strong wooden blocks placed in pockets cut in opposite sides of a passage, the blocks being set at right angles to the axis of the girder. Cast-steel shoes are sometimes used. In weak ground girders may be grouped in threes or fours and wedged on wooden bars or sills. Ordinarily girders are spaced 3 to 4 feet apart, uniform spacing being best.

Use of cambered girders in mines (England). Sci. and Art of Min., January 23, 1932, p. 210. The use of camber girders or "springs" in English coal mines is common practice, particularly where there is rock movement or in "motive" zones near the coal face. Camber girders are simply girders bent to a curve and set from the walls instead of the floor. They take the place of the old-fashioned, but effective, herringbone strutting, but being of steel and without joints, are very much stronger. They allow the road to settle naturally with the adjacent strata and at the same time support the roof and opposing side squeeze.

Use of cambered girders in subsiding area. Trans. Inst. of Min. Eng., London, vol. 82, p. 344. As settlement of roof takes place the cambered girder settles also, whereas the arch girder pushes up into the roof, often breaking what would otherwise be a sound top. It is much easier to make repairs by taking up the floor than to remove arch girders and reset them after reripping.

Use of steel arches in mines (England). Sci. and Art of Min., January 23, 1932, p. 210. The use of "horseshoe" arches of rolled steel, commonly known as "rings" are coming in general use, one mining division having 73 miles of roadway supported by them, while another division had about 350 miles of roadways supported by steel rings in 1929.

Arches bent and distorted by rock pressure can be straightened and put in use again with apparently as good results as when new. The withdrawal is also readily accomplished, as is shown by an instance where 6-foot arches were withdrawn from a roadway 600 yards long at the rate of 12 arches per shift of two men.

An E arch with its accessories consists of 2 half arches, 2 fishplates, 4 bolts and nuts, 36 struts of round timber, 6 notched struts, 2 sole pieces, 2 clips with bolts and washers.

The support of junctions by steel arches. Fowkes, Alfred, Iron and Coal Trades Rev., London, February 5, 1932, pp. 233-234. Steel arches have been found very satisfactory for the support of ground which had formerly proved difficult when braced with girders and props; however, their use has been confined largely to coal mines in England and on the continent. The concentration of weight at the junction of roadways causes trouble due to change in sequence of supports, and distortion of the girdered junctions occurrs. Various types of steel structures were tried before the present form of support was adopted. Arches made of 8-foot steel are now carried as close to the edge of the junction as possible and two pairs of the ordinary standard 12-foot girders are bridged diagonally across the corners of the junction, one pair passing under the other; the legs of the 12-foot arches abut closely to the 8-foot arches, the whole of the arches forming a complete anchorage with each other and present the appearance of a vaulted ceiling supported by columns.

In order to get the diagonal arches, one above the other, it is necessary either (1) to set one pair of diagonals upon an extra foot block or (2) to cut approximately 6 inches off the ends of the legs of one pair of diagonals, the second method has proved the more successful, as it keeps the whole system of arching uniform.

The support of underground roads by steel arches. Coll. Guard., November 27, 1931, pp. 1790-1792; December 4, 1931, pp. 1880-1882. The use of steel arches for the support of mine roads in the Yorkshire coal mines is being rapidly extended. Considerable skill is required to erect the arches due to a wide difference in character of the roof, floor, and sides of the roadways. Three types of arches are in use - horseshoe, straight-sided, and splay-legged--which generally conform to standard sizes. Details are given as to size and weight of the various types used. Problems that require special consideration and study in the application of the arches are: (1) Subsidence; (2) the development of pressure upon roof and side supports; (3) stability of roof and sides so long as supports are not disturbed. The special conditions under which the types of arches are applicable are outlined. The protective devices employed with steel arches are: (1) Lateral support between arches to prevent sidewise buckling; (2) lagging and packing to prevent radial distortion in the plane of the arch; (3) foot blocks to provide a yielding foundation; (4) correct spacing of arches.

The use of steel in shaft lining. Eaton, Lucien, Eng. and Min. Jour., February 1932, p. 86. The advantages of steel as a material for shaft sets are as follows: (1) It takes less room than wood; (2) it is strong, bends more often than it breaks, when bent can often be straightened on surface and can be used over again; (3) it is easily placed; (4) it is fireproof; (5) it does not rot and is not subject to the attack of insects. Its disadvantages are: (1) It often costs more than wood; (2) when wrecked through accident or excessive pressure, it can not be chopped out, and it can not be cut to fit underground; (3) it is corroded by acid water.

Corrosion of iron and steel. Iron and Steel of Canada, December, 1931, p. 186. The American Society for Testing Materials has recently issued a report by Committee A-5, on the corrosion of iron and steel, which gives the result of extensive tests of metallic coatings and inspections of galvanized sheets, hardware, structural shapes, conduits, and so forth, showing failure of a number of coatings. This work should be of interest to the mining engineer who has to do with iron and steel structures both on the surface and underground.

USE OF CONCRETE AS SUPPORT

Concrete supports in the Rand mines, South Africa. Min. and Ind. Mag., South Africa, March 2, 1932, p. 29. Concrete is usually stronger than the rock or ore in a mine and where properly placed constitutes a strong support. Concrete was first used in the mines as monolithic columns and later as built-up columns, "disks" or "pancakes" of concrete being superimposed to form the support. The disks have the advantage that they can be made on the surface. When the columns have a soft formation as a footing and have a capping of timber they give remarkable good service.

The disks range from 27 to 33 inches in diameter and are 3 to 4½ inches thick, with a central 4-inch hole. They are reinforced by wire rope. The concrete mixture is 1 of cement to 3 of sand and 6 of 1½-inch-mesh rock. Uniform mixing and treatment in a "pickle" bath for a week insure excellent disks.

The advantages of concrete over pigsties are: (1) Less risk of fire, (2) they become harder and stronger with age, (3) stopes can be kept cleaner, (4) they are easily erected, and (5) supervision is made easy.

The Schaefer lining for roadway. Coll. Guard., London, April 15, 1932, pp. 734-735. The Schaefer concrete lining has been employed in the English coal mines, although up to the present time to a rather limited extent. The lining consists of a series of concrete arches abutting one against another with or without inverts and suited to the particular conditions. The arches are constructed of well-matured concrete blocks in the form of the letter T, and with a proper taper to permit close contact of the faces, as they are built dry. Through holes cast in the rib of the blocks, steel rope is passed

as reinforcement, and is secured to the blocks by grouting as the lining is erected. The invert is first constructed and the arch is built on steel centers which are advanced as the rings are completed. Wooden compression courses are inserted where pressure is expected to develop.

Larger excavations are required for these linings than with most other methods of support, and they are more difficult to erect. Furthermore, their success as elements of support depends largely on the careful packing of a considerable quantity of small-sized waste back of the lining. Continuous Schaefer lining is costly, for which reason the rings forming the lining should be spaced, thus making it comparative with steel arching. One of the merits claimed for the Schaefer lining is that each ring can move independently.

Use of concrete for shaft lining. Eaton, Lucien, Eng. and Min. Jour., February 1932, pp. 86-87. The use of concrete as a lining for shafts has increased rapidly, owing to strength and increased facility in handling and placing. The advantages of concrete are as follows: (1) It is fireproof; (2) its strength increases with age; (3) it does not rot, nor is it corroded or easily disintegrated by acid water, although this does sometimes occur; (4) it may be poured in place, making a continuous support; and (5) it may be used as precast blocks, which afford greater uniformity in strength.

Cementation in shaft sinking. Detchon, R. J., Sci. and Art of Min., January 9, 1932, pp. 194-195. The author describes a method of cementation used in shaft sinking that came under his observation, where water occurred under pressure. Boreholes were put down near the edge of the shaft and as truly vertical as possible, approximately 100 feet deep. The boreholes, four in number and 2½ inches in diameter, formed a square within the shaft, being an equal distance apart and within 2 feet of the shaft sides. Short lengths of 2-inch pipe, 2 feet 6 inches to 3 feet long, were inserted in the boreholes with threaded ends upward, and wedged tight with soft wood plugs or wedges, thus sealing the water off and allowing it to discharge through the pipes into the shaft. Valves were fitted to the ends of the pipes and the pipes connected with the pumps circulating the cement mixture.

The cement was mixed in a barrel in the proportion of 95 parts of water to 5 parts of cement, or a ratio of 19:1, to form an intimately mixed liquid cement. The pump used in handling the cement had a 9-inch air cylinder, a 12-inch stroke, and a 2-inch duplex plunger. It was capable of developing a pressure of 1,600 pounds per square inch. At shallow depths the pressure of cementation was kept below that due to the head of water at the depth of the shaft, in order to prevent loss of cement through escape in fissures.

Tunnel grouting at Cobble Mountain. Hatch, Harry H., Eng. News-Rec., December 31, 1931, pp. 1037-1039. Voids formed or left behind concrete linings in rock tunnels require grouting to fill them and insure good contact and support. The maximum opening occurs at the crown and narrows to a minimum at the spring line; there are also irregular open spaces in the rock face. Drain pipes were placed at water seams and other sources of water, and drains were

also placed in the lining to take care of running water. Grout holes were placed after the lining was complete, and were 2 inches in diameter and fitted with 1½-inch pipe. Three types of grouting were employed according to placing of holes: (1) A single line of holes spaced at various intervals in the roof of the tunnel; (2) a hole placed on each haunch of the arch and located 30° above the spring line; (3) eight holes drilled 10 feet through the lining and into the rock, at 45° intervals around the section, beginning at the crown.

The grouting machine consisted of a small cylindrical tank with a flap gate at the top for charging materials; air was admitted at the bottom. A maximum air pressure of 100 pounds per square inch was used in placing the grouting. Following the initial charge of neat grout, more sand was added, but never more than 2 parts of sand to 1 of cement. Other details are given relative to practice under variable conditions, amount of grout used, movement of grout, check on filling of voids, and so forth. Costs are also given.

Cementation of cast-iron tubbing at Shireoaks colliery. Walters, L. Ward, Trans. Inst. of Min. Eng., London, vol. 82, part 4, pp. 320-331. A detailed description with sketches and photographs is given of cementation of corroded shaft tubbing in an English colliery. The tubbing was lined with 3-inch brickwork which adhering to the defective metal tubbing and caused openings to form when the brick was removed in the search for leaks and in preparation for cementation. Extensive breaches would cause a rush of water and reverse the ventilating currents. Furthermore, owing to the weakened condition of the tubbing, no great pressure could be put upon it. It was then decided to cement one section of the tubbing at a time, and ultimately to complete the whole length so as to form a continuous lining.

In order to prevent any excessive pressure on the tubbing, a T was put in the pressure main between the cement pump and the shaft. From this a vertical pipe was run for 15 feet and then carried back across to the mixing tank. When cement would not enter the tubbing, the pump would force it back into the mixing tank; the only extra pressure then imposed would be that due to the 15-foot head. The tubbing was tapped by drilling through the original plug holes, so that 1¼-inch nipples and provided with valves could be inserted. Flexible connections joined the nipples with the pump discharge circulating the cement mixture. No serious difficulty was experienced in the work, which was done satisfactorily and with no enforced delay.

The use of gunite in fractured and loose ground. Smith, T. E., Eng. and Min. Jour., March, 1932, p. 153. Gunite has been used successfully in various mines to hold together fractured and loose ground. The author describes the methods employed in the Butte mines which have proved satisfactory. When retimbering drifts through loose, blocky ground, gunite is used to prevent breaking down and running. The side lagging is removed and gunite is distributed over the exposed rock surface and in the cracks and interstices that can be reached. After three or four hours, the back lagging is removed and

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the loose material is cleared away. After the removal of back lagging and loose ground, all cracks are filled with a stiff mixture and all exposed rock surfaces are covered with 1 inch of gunite. A stiff gunite is more suitable than a thin gunite for cracks that can be reached readily, while thin gunite will penetrate gractured ground that is more inaccessible. A relatively dry mixture should be used for coating exposed rock surfaces.

Gunite filling will steady loose ground better than numerous sprags and stulls; also, an important advantage in the use of gunite is the warning given by formation of cracks.

About three cars of gunite material, including 1,000 pounds of sand and three bags of cement, are required per set, two men performing the work of changing sets in two days' time.

Underground air-drying pockets. Alan Wood mine, Dover, N. J., Min. and Met., November, 1931, p. 492. A description is given of the construction of an underground ore pocket for handling wet ore prior to its delivery to the mill. The cementation of the rock walls in order to keep out water was accomplished by the Francois method. A lining from 5 to 10 feet thick was provided without decreasing the size of the pocket as cut in the rock; that is, the lining was formed by grouting the rock walls. The cost of cementation was reduced by two-thirds and the time required by one-half. The rock is gneissoid granite.

USE OF WASTE PACKS AND FILLING

Packing waste in coal mines. Brass, T. F. S., Trans. Inst. of Min. Eng., London, November, 1931, pp. 149-152. The support of underground workings by the use of waste filling is of such importance that information relative to its application in all mines is of interest to the mining engineer. The methods of packing waste in the Continental coal field in Europe are as follows: (1) Hand-packing; (2) pneumatic packing; (3) blowing the dirt from a conveyor into the gob; (4) dummy-road packing; and (5) partial packing. The source of the packing material is from driving, ripping, dirt in seams, washing plants, and surface refuse materials.

Hand packing is costly in transport and placing. It is delivered by cars, conveyors, and chutes and put in place by hand. If coarse, it is built into walls, whereas the while small material is confined in inclosures formed by wire netting. Pneumatic packing is under 3 inches in size and is discharged through bunker and pipe under air pressure of 3 to 8 pounds per square inch. The packing is conveyed about 300 yards, beginning at the foot and advancing upward by removing sections of pipe. Eight yards are packed for each position. About 200 cubic feet of free air are used per cubic foot packed. When packing is blown in place, it is fed to a conveyor, from which it is discharged onto an inclined plate and thence fed into a tube and blown into the gob. No pack walls are built, as the material is stowed in a damp condition. Dummy-road packing is done in beds under 36 inches in thickness,

the dirt being obtained by cutting dummy gates in the face and packing it solid between the gates. In the partial-packing method, chocks or packs of fallen waste are built across the mouth of each room and are reset as the face is advanced. All timber is withdrawn from the gob, the packs and fallen top rock forming the support.

Stone packs in the Rand mines, South Africa. Min. and Ind. Mag., South Africa, March 2, 1932, p. 29. Stone packs are cheap, as the material used is at hand; even ore may be used to be reclaimed later. The outer walls of the packs are strongly and evenly built to prevent collapse under pressure of the hanging wall. The walls may be reinforced by old rails, pipes, and timber laid crosswise. Packs are effective for dips up to 35°; they are built from below upward, are supported on the lower side by poles, and may serve to fill in between other supports such as concrete and pigsties.

The Sloan wire pack has proved very effective under certain conditions, the advantages being strength, lightness, the ability to hold the contained rock under compression, and cheapness of construction. The pack is made of 6-inch mesh wire netting which incloses a mass of rock or may encircle a pillar that is failing and reinforced by rock filling. The more the netting is compressed and reduced in width, the stronger the pack becomes.

Rock bursts in coal mines and effect of stowing. Bracht, _, Marx, _., and Spackeler, _., Ztschr. Berg-Hütten und Salinenwesen, vol. 79, 1931; Coll. Guard., January 8, 1932, p. 72. All rock bursts present certain concurrent and characteristic features, and may therefore be assumed to have a common cause. There are two kinds of bursts - stress and roof. The former are caused by differences in compressive stresses as governed by depth and nature of workings. Heavy roof bursts are caused by flexure of the solid main roof, leading to sudden breakage, under the considerable forces released by elastic deformation. The shock is transmitted to the rock adjacent to the working face, but at a considerable distance from the seat of the disturbance a wide variety of reactions may occur.

Stowing has frequently been suggested as a remedy for rock bursts, but has not proved satisfactory. Stowing can not prevent solid roof strata from bending or sagging up to the dangerous fracture zone, but it can retard the effect. The result of stowing may be that roof bursts will occur more seldom, but with greater intensity.

Disadvantages in the use of waste filling. Bull. Inst. Min. and Met., London, March, 1932, p. 4. Some of the disadvantages in the use of a filling of waste-rock ranging in size from 8 to 10 inches (about 80 pounds in weight) to plus 1/8-inch, are given for one instance as follows: (1) The stopes had to be timbered as worked, the ground being heavy though hard; (2) all timber had to be removed as filling was placed in order to avoid fires; (3) a minimum of dilution of the waste rock was permitted; (4) water admitted to the mine soon became acid and was costly to pump; (5) the ore beneath the waste filling had to be removed up to the filling overhead.

The rock used as filling, in this instance, was siliceous porphyry, and had to be loaded from shafts, trammed, dumped, built into walls, and shoveled into place. The specific gravity of the ore was about 4.8; that is, every cubic meter of void made produced 4.8 tons of ore theoretically. Owing to settlement of the ore only 3.25 to 4.25 tons were obtained for each cubic meter of filling placed. The amount of filling placed per shift varied from $1\frac{3}{4}$ to $3\frac{1}{2}$ cubic meters.

Support in the Mysore gold mines, India. Iron and Coal Trades Rev., April 8, 1932, p. 593. Owing to the variable width of lode, 1 inch to 30 feet, and the variable dip, horizontal to vertical, but usually about 65°, props and pack walls are commonly employed. Props are set securely into the walls, 10 to 12 feet below the level and usually three to a stope. On the props are placed poles forming a platform upon which is built the foundation wall about 2 feet thick. Side walls are next built up from the bottom wall to the level above, the inclosed space being filled with small-sized waste rock.

In order to hold the side walls in place, small wedge-shaped pieces are driven into spaces between the rock foot and hanging walls and the pack walls, thus fastening them in place so that the supporting props may be removed. The pack walls are thus built in sections from above downward and all timber is removed. As stoping is continued along the lode, other packs are erected at varying intervals. The worked out stopes are in this manner kept open for the passage of men and air currents.

Sand filling in the Rand mines, South Africa. Min. and Ind. Mag., South Africa, March 2, 1932, pp. 29-30. The use of sand filling has advanced from a minor rôle, such as that of supplementing pillars, to a major and standard form of support in the Rand mines. The sand is made into a light pulp by adding water and is then delivered by pumps to dewatering cones placed over the boreholes, through which it is conveyed to the mine workings. The cones return the overflow water to the sand tanks and separate out a certain amount of sand that it is desirable to keep out of the mines. Where inclined shafts are available, the sand is conveyed underground in launders. Where boreholes are employed they are usually 7 to 11 inches in diameter.

It is desirable to have a number of feeding points for proper distribution underground. Occasionally the sand has to be pumped to the boreholes and for considerable distances, often necessitating two lifts, owing to rising ground. Sand to the amount of 2,000 to 3,000 tons is placed underground daily by such a system.

In sand filling, a dip of 12° barely serves to move the material from the dewatering cones when 30 per cent of moisture is present. With a lower inclination more water must be added. A minimum amount of water should be used, as a surplus adds to the danger of loss of control and robs the sand of its supporting power. A three-walled chamber is erected in a stope at

the location of a borehole, and from the down-dip side a launder conveys the sand to the points desired. Particular care must be taken in the preparation of an area for sand filling. Wire netting and cocoanut matting are often used to aid in strengthening and rendering tight the confining barricades.

The blast system of goaf stowage. Deuschl, E., Glückauf, July 4 and 11, 1931; Coll. Guard., November 6, 1931, p. 1537. The best high-pressure process is that developed by the Torkret Gesellschaft, in which the waste is fed into an upper chamber completely sealed off from a lower one. From the lower chamber the material is fed into the compressed-air stream by means of a bucket wheel rotating on a vertical axis. A pipe conveys the waste to the storage place.

The "cell-wheel" stower, also largely used, consists of a cylindrical casing in which a 6-cell wheel revolves on a horizontal axis. The cell-wheel serves as the distributor to the air main, the waste passing through it to the pipe below.

The main difference in the two systems is that the bucket wheel is subjected to a uniform pressure on all sides, while in the cell-wheel system the air pressure acts upward only. These machines are extensively used and handle the waste material successfully.

Pneumatic stowage of goaf. Poole, Granville, and Whitton, J. T., Mech. Handling, November, 1931, pp. 347-353 (reprint from Coll. Eng.). The objections to hydraulic stowage are partly overcome by the use of pneumatic stowage; they are as follows: (1) It is impossible to fill the void completely; (2) stowage in flat seams is more difficult than in an inclined seam; (3) drainage is made more difficult, including clarification and pumping of water; and (4) the effect on bottom and walls of workings is injurious.

Results of experiments with sand, sand and washery débris, and washing débris alone are given, with advantages and disadvantages of each.

Subsidence with different methods of filling. Mech. Handling, November, 1931, pp. 347-348. The following comparative figures give the effect on subsidence of the surface when using different methods and kinds of filling: (1) Hand stowage with material from 0 to 12 inches, subsidence 40 to 50 per cent; (2) mechanical stowage with material from 0 to 3 inches, subsidence 15 per cent; (3) pneumatic stowage at a pressure of 3 to 6 pounds per square inch, subsidence 25 per cent; (4) pneumatic stowage at a pressure of 15 to 22.5 pounds per square inch, subsidence 15 per cent.

Sand-filling methods at Hodbarrow (haematite) mines, South Cumberland. Jones, A. Alec, Bull. Inst. Min. and Met. Eng., London, February, 1932, pp. 1-19. The use of sandfilling of workings was to prevent water and water-laden sediments from entering the workings. The filling material is sand, readily accessible in unlimited quantities on the surface. The means of conveying the

sand underground was by shafts and boreholes. At the beginning of the work the sand was sluiced into the boreholes, the pipes lining the boreholes being cut off from time to time to permit ready movement of sand and water. A bunker or pit was excavated at the shaft location into which sand was dumped and from which it was sluiced into the mine.

Launders were first used for distributing sand underground, but the limit was soon reached for this form of gravity filling, as there was not sufficient inclination for the launders. The next development was to make use of the momentum of the descent in boreholes and shaft. Cast-iron pipes 4 inches in diameter were employed, fitted with Victaulic joints and with a gradient of 1° from the horizontal. While there was a decided improvement in radius of operation, the volume of sand delivered was not improved, so that filling could not keep pace with ore extraction.

Compressed-air agitation was then tried and adopted. Compressed air was introduced into the delivery line at a pressure of 60 to 70 pounds per square inch. The injector finally adopted minimized the sand blasting of the pipe by injecting the air at a bend and along the axis of the delivery pipe. Injectors were also introduced at points in the delivery pipe in order to aërate the mixture and prevent choking of pipes. Furthermore, by aërating the feed a mixture containing a minimum of water is possible, the feed having less than 50 per cent of sand, by volume; up-grade deliveries are then possible. The longest pipe line was 300 yards and had many bends, with up-gradients of about 1 in 100, with three injectors more or less evenly spaced.

Continental mines employing "hydraulic stowing." Taken from paper on "Sand-Filling Methods at Hodbarrow, South Cumberland." Jones, A. A., Bull. Inst. Min. and Met., London, February, 1932, p. 19.

The following mines employ hydraulic stowing: Petite Rosselle, Forbach, Alsace, using a mixture of sand, burnt slate, boiler ash, and flue dust.

Velsen mine, Saarbruck, using the product of a sandstone quarry, the pieces not exceeding 2 inches in diameter.

Zwickau, German Silesia, using sand from a sand pit mixed with tailings.

The Concordia pit, German Silesia, using granulated stag mixed with sand.

Mystowitz, Poland, using sand excavated from sand pit.

Villeboeuf mine, St. Etienne, France, using a mixture of crushed conglomerate rock, cinders, with mine and washery refuse.

Among others, the following mines also use hydraulic stowing in some form or another: The Ludvigsgluck Colliery of Upper Silica; the Salzer-Neuack Colliery of Essen, at Centre de Jumet, St. Louis, Belgium; and the Gewerkschaft mine of Thyssen.

Sand filling through pipes. Eaton, Lucien, Eng. and Min. Jour., March, 1932, pp. 137-138. Sand has decided advantages in filling for support in mine workings, such as slight compressibility and ease and economy of placing. The relative compressibility of sand and coarse material is 5 and 10 to 25 per cent. The finer the material used, the more complete the fill in underground excavations.

The methods employed in sand filling are: (1) By hand; (2) by water; (3) by compressed air, and (4) by air and water combined. When filling is done by hand, the sand is transported in the dry state in cars, dumped in raises, and distributed by cars, wheelbarrows, or scrapers. When water is used in handling and distributing sand, it is flushed down pipes or through boreholes, and distributed by pipes or launders. The material distributed may range from 20 to 75 per cent in solids. When compressed air is employed, the dry sand is introduced into the mine through pipes and distributed by an intermittent, high-pressure system through a receiving tank and discharge pipes, or by a continuous low-pressure system, in which the sand is fed into a pipe with a large volume of air. When compressed air and water are combined, the sand is flushed through pipes or boreholes and distributed through pipes with small jets of air introduced at intervals.

The advantages and disadvantages of the various methods are given with relative economies in their use.

Cement-gun control of loose and moving ground. Smith, T. E., Eng. and Min. Jour., March, 1932, pp. 152-153. The use of monolithic concrete structures have not proved successful as they crack and break up when subjected to pressure. In the Butte mines it was found necessary to gunite the failing concrete stoppings every two weeks. A method developed in 1930 to use sand as a filling for dams and stoppings has proved highly successful, and was first employed to isolate abandoned areas in the upper levels of the Leonard mine.

A bulkhead is first built with old timber across the drift to be closed, and after being wedged tightly to the sides and back, the loose ground all around the drift is removed. A second bulkhead or dam is erected 6 feet back of the first and to a height of 3 or 4 feet. The space between the two is then filled with sand, which is packed with water while being placed. The sand may be placed with a cement gun, which packs it tightly. The inner bulkhead is gradually raised until it nears the top of the drift, when the sand is shot through a narrow opening at the top. Any movement of the rock tends to pack the sand more completely.

When delivering sand through a cement gun, variations of pressure, rate of feeding, or flow of water are not important. By moving the nozzle steadily to and fro, the sand can be placed in a homogeneous mass. When delivered by a cement gun, the sand will stand vertically if 20 per cent or more of the material will pass a 30-mesh screen, and if the correct pressure and amount of water is used.

Sand filling on the Rand. Bull. Trans. Inst. Min. and Met., London, March, 1932, pp. 8-10. Sand filling as employed on the Rand is not to support the top rock but to enable the ore to be taken out completely. The method of working is by a kind of stall-and-pillar system. The dip is slight, being close to 14° to 15°, and pillars are left to support the roof, but as they are valuable it is desirable that they be mined. With this end in view, large areas are filled subsequent to extraction. The filling does not go on contemporaneously with the mining, but afterwards the mined areas are filled on a large scale, and then the pillars can be removed. In certain instances, to get at the pillars, it has been necessary actually to dig through the sand which has become consolidated by the great pressure to which it is subjected.

In the New Modderfontein the stopes are wide and the dip is slight. The upper portion of the mine was near the surface, and support was required to prevent caving, which would have admitted surface water to the mine. Hundreds of thousands of tons of sand had been lowered into the mine through boreholes and packed much better than other forms of filling which were tried.

The advantages of hydraulic sand filling in metal mines. Bull. Inst. Min. and Met., London, March, 1932, p. 7. The use of sand filling is of such importance as an aid to mining that its rapid extension is ensured where conditions are favorable to its application. The more important of the advantages are: (1) The possibility of completely filling large areas, as compared with the relatively small areas which can be packed with dry-filling; (2) the possibility of reaching areas which are partly caved and inaccessible; (3) additional safety to underground workers when the fill is carried out in current workings; and (4) the improvement of ventilation owing to old workings being completely shut off, and the prevention of fire owing to old timber being buried.

Rubber pipe lining (for filling stopes). Homer, D. D., Eng. and Min. Jour., October 26, 1931, pp. 367-368. Rubber-lined pipe for handling and distributing mill tailing as filling for stopes is described. Standard steel pipe 3 inches in diameter and 12 feet long is used, in which is placed a $\frac{1}{4}$-inch thick, vulcanized rubber lining. Special attention must be given to the connections; otherwise the sand will wear a passage between the lining and the pipe.

A 3-inch pipe has a capacity of 35 tons per hour, the cost per ton of sand being about 20 cents.

I. C. 6652

October, 1932

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

MINING METHODS AND COSTS AT THE HART SPUR PIT OF THE FORT WORTH SAND & GRAVEL CO. (INC.), FORT WORTH, TEX.

BY

THOMAS E. POPPLEWELL

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INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AND COSTS AT THE HART SPUR PIT OF THE FORT WORTH SAND & GRAVEL CO. (INC.), FORT WORTH, TEX.[1]

By Thomas E. Popplewell[2]

INTRODUCTION

This paper is one of a series being prepared for and published by the United States Bureau of Mines describing the mining and treatment methods and costs obtained in sand and gravel plants throughout the United States.

These papers are designed to disseminate technical information regarding the methods used. The cost tabulations represent operating expenditures only and not total costs. It is recognized that publication of total costs might in many instances cause embarrassment to individual operators as well as to the industry as a whole. On the other hand operating costs are essential to the technical discussion and study of the methods employed. The attention of the reader is specifically called to this differentiation in order that no misunderstanding of the scope of the cost tabulations shall ensue.

The methods employed at the Hart Spur pit of the Fort Worth Sand & Gravel Co. are described in this paper. The pits are located along the Trinity River, from Fort Worth eastward to the Tarrant County line, and are adjacent to the Rock Island Railway Co.'s tracks, to which the standard-gage gravel-pit track is connected at Hart Spur.

ACKNOWLEDGMENTS

The writer wishes to acknowledge the assistance of the Bureau of Economic Geology of the University of Texas and the suggestions of J. R. Thoenen of the United States Bureau of Mines, in the compilation of this report.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6652."

2 One of the consulting engineers, U. S. Bureau of Mines, and secretary, Fort Worth Sand and Gravel Co. (Inc.).

HISTORY

When sand and gravel were first used for building roads and concrete structures in this vicinity, the pits were operated by hand and deliveries were made by wagon from pits that are now within the city limits of Fort Worth. The company is still working a number of these pits as one unit, although its operations now extend down the Trinity River about 15 miles. In 1919, the company opened the first machine-operated gravel pit in the county and produced pit-run concrete gravel for the building trade. In the following year it built a small gravity washing and screening plant in order to make a product that would compete with crushed stone. The progress made in the design and control of concrete for specific purposes has created such a demand for better graded and washed materials that the company has been forced to remodel old plants and to build additional ones of modern type.

GEOLOGY

Two types of gravel deposits are found in Tarrant County: (a) River gravel, which is found in the lowlands and consists of rounded clean pebbles and sand with little or no cementing material; and (b) pit gravel, which is found in the upland or earlier river bottoms and consists of angular, flat, elongated particles mixed with clay and other foreign matter and particularly with plastic clay balls. The surface over both types of deposits is generally level. The upland gravel deposits are usually the thicker and have heavier overburden; otherwise the two types of deposits are quite similar. These deposits of sand and gravel were washed down the Trinity River from disintegrated medium-hard limestones upstream. As might be expected the larger-sized pebbles were deposited first and the smaller sizes were washed on downstream. The deposition of the material seems to have been caused by the interference of small tributaries, as the deposits are almost invariably found on the upstream side and immediately adjoining the mouths of the small streams. (See fig. 1.) The deposits vary in size from a few acres to over a hundred acres and exist mostly on the north side of the river - a fact which has never been satisfactorily explained.

By mining upland and river gravels at the same time with separate draglines the two can be mixed in the washing plant with good results. A sandy soil and clay varying in thickness from 3 to 8 feet coveres the deposits, which vary from 8 to 25 feet in thickness. In the river deposits water is found from 6 to 8 feet below the surface. This enables the operator to produce a relatively clean pit-run gravel, containing only about 1 per cent clay and silt. Some deposits have a natural grading which permits the use of this pit-run material in making concrete for small jobs, such as residence foundations, sidewalks, and street-paving base where asphalt topping is applied. The deposits are rather free from conglomerate (particles of sand and gravel cemented with free lime from the pit water) except at the water level.

The deposit at Hart Spur consists largely of river gravel, but there are some adjacent upland gravels. At the time of this report the river gravel only was worked.

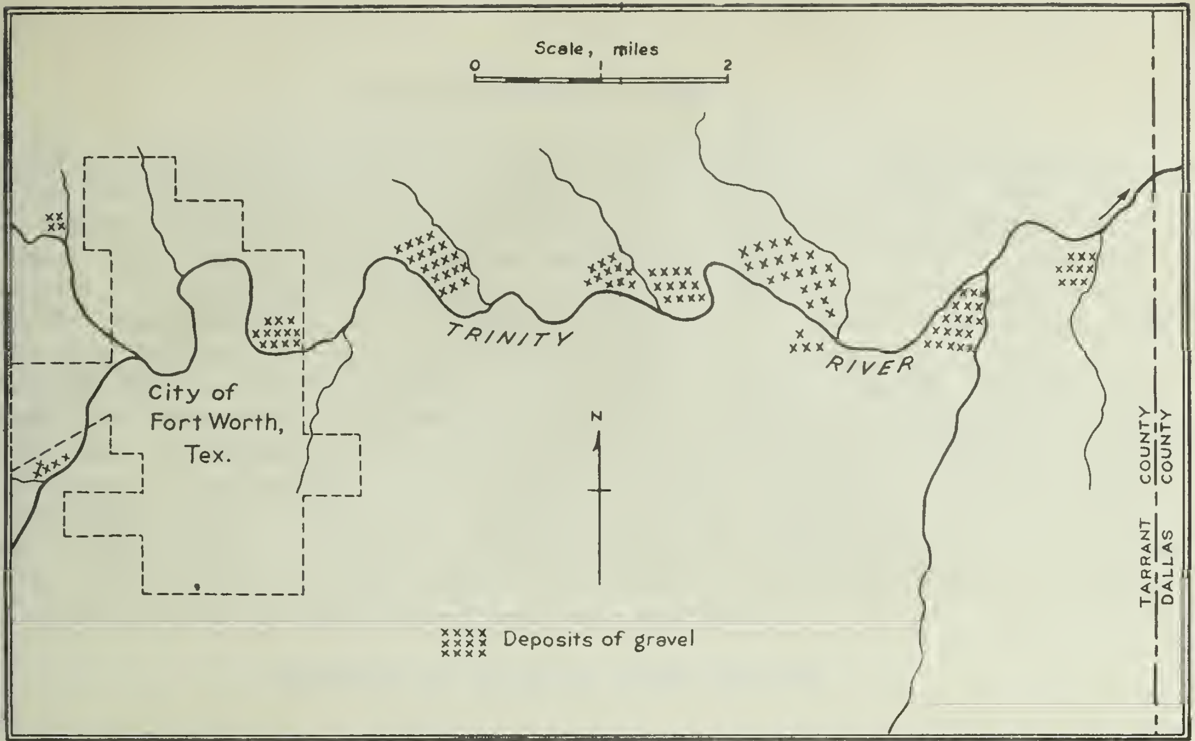

Figure 1.- Sketch showing gravel deposits along the Trinity River, Tex.

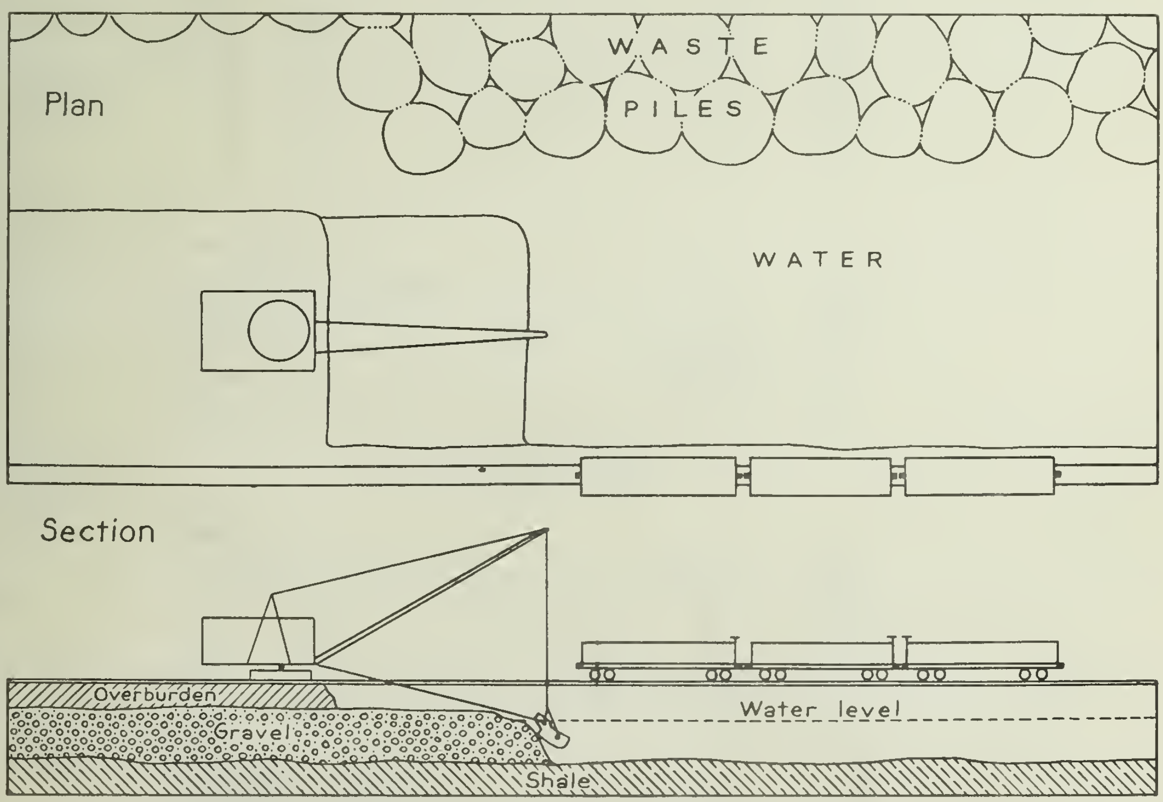

Figure 2.- Plan and section of typical pit operation

PHYSICAL CHARACTERISTICS

Under the top soil which is about 2 feet deep, clay 3 to 8 feet thick overlies the gravel deposit. This overburden forms about 40 per cent of the total material handled, and is cast back into the pit made by previous excavations. The sand and gravel bearing stratum is from 8 to 25 feet deep and is usually underlaid by blue shale. The gravel deposits vary in size and shape, but the average size of commercial deposits is about 70 acres and they are usually triangular in shape - a fact that is accounted for by the deposition of the material on the upper side of the tributaries and between them and the main river. The material is loose and runs approximately 40 per cent of sand and 60 per cent of gravel. The sand and smaller sizes of gravel are siliceous, whereas the larger gravel is mostly limestone. Structurally the sand and gravel is sound, but occasional flat, elongated "floaters" are found. These floaters amount to about 1 per cent of the gravel in the upland deposit, and being soft they increase the amount of abrasion shown by tests. The floaters found in the river gravel are negligible.

ANALYSIS OF MATERIAL AFTER WASHING

Limestone gravel

Retained on:		
3-inch screen	per cent	0.00
2 do.	do	2.1
1½ do	do	5.4
1¼ do	do	9.4
1 do	do	21.1
¾ do	do	48.6
½ do	do	81.1
¼ do	do	98.0
Passing:		
¼-inch screen	do	2.0
Specific gravity		2.61
Dry weight per cubic foot ...	pounds	96
Voids	per cent	41
Abrasion	do	12

Siliceous sand

Retained on:		
4-mesh screen	per cent	3.0
8 do	do	25.0
14 do	do	40.0
28 do	do	54.0
50 do	do	90.0
100 do	do	99.5
Passing:		
100-mesh screen	do	.5
Specific gravity		2.65
Dry weight per cubic foot ...	pounds	105
Voids	per cent	36

PROSPECTING, EXPLORATION, AND SAMPLING

Prospecting and exploration is begun in areas near outcrops of a gravel-bearing stratum, usually in creek banks and in the bottom lands near willow trees, which commonly grow above the gravels in this section. The first testing is done with a regular 8-inch post-hole auger, fitted with an extension handle so that a depth of 10 or 12 feet can be reached. This preliminary work does not indicate the quantity or the quality of the material or the depth of the gravel stratum, but it serves to outline the deposit and determine how much overburden must be removed. Sometimes holes are sunk more or less at random, but they are generally spaced in rows at right angles and distanced roughly in multiples of 100 feet.

If the results of the preliminary investigation justify it, the area to be tested is staked off in checkerboard fashion at intervals of 100 feet. At every third stake a test pit $3\frac{1}{2}$ by 8 feet is dug to the top of the gravel stratum and is continued as a round hole about $3\frac{1}{2}$ feet in diameter to the bottom of the gravel or until water is struck which prevents sinking further by hand. It is not important to reach the bottom of gravels below the water level because experience has shown that the material under water is better than that above it. However, if the gravel stratum above the water is thin it becomes necessary to sink through the gravel in order to estimate the quantity of material in the deposit. A small orange-peel bucket is used inside a 20-inch casing to continue the digging below water level. This bucket is handled on a small crane and is operated by hand. However, if there is as much as 6 feet of gravel above water, the hole is deepened by hand for 2 feet below water level and the excavation is stopped. After reaching a profitable depth it is not important to determine lower extent of the gravel, for only an approximate average bottom can be determined at best, as it will often dip or rise irregularly in the 300 feet between pits.

The test pits are platted, showing the depths of overburden and of gravel, and from this data the extent and approximate yardage of the deposit is calculated. Notations are made on the plat showing the quality of material in each test pit.

The overburden is piled to one side of each test pit and kept separate from the gravel. If the character of the gravel changes from top to bottom, then the different kinds of material are piled separately so that a sample can be taken from each pile. A screen analysis is made of each of these samples, which, after being separated into gravel and sand, are mixed in proportion to the thickness they represent. This composite sample is sent to the State Testing Laboratory for a report on the apparent specific gravity, abrasion, absorption, voids, screen analysis and weight per cubic foot. The results furnished by the State Laboratory are checked by the company and if there is much variation an independent testing laboratory is consulted and further sampling of the deposit may be undertaken.

The walking-type dragline is powered by a 120-hp., 2-cylinder Diesel engine with 16-inch stroke and bore, operating at 260 r.p.m. and equipped with a 70-foot boom and a 3-cubic yard bucket. It has 110 feet of $1\frac{1}{4}$-inch, plough-steel, 6-strand, 19-wire Lang-lay drag cable with wire-rope center. The hoist cable is 220 feet long, 7/8-inch diameter, 6-strand, 19-wire, plough-steel, regular-lay with hemp center.

This machine operated 4,200 hours during 1931 and consumed 34,224 gallons of distillate, or 0.10 gallon per ton of sand and gravel mined. The distillate for this machine is pumped from a special tank mounted on top of the locomotive tender. The price as of January 1932 was $2\frac{1}{2}$ cents per gallon. The operating cost per ton with this machine is considerably less than with the steam machine and for that reason it is used more. At present, only one of these machines is used, but during the height of summer business both machines are operated on the river deposit.

HAULAGE

After being stripped, the sand and gravel are loaded into railroad cars and hauled to the washing plant by steam locomotives. Two 90-ton locomotives pull empty cars from the railroad to the dragline, a distance of $3\frac{1}{2}$ miles. After the cars are loaded they are hauled $2\frac{1}{2}$ miles over a level track to the washing plant, where they are emptied and refilled with washed material. They are then hauled 1 mile up a $\frac{1}{2}$ per cent grade to the main line of the railroad for delivery to the consumer. A third locomotive, weighing 80 tons, is kept in reserve to replace either of the others when laid up for repairs. During the year the locomotives worked 7,380 hours and consumed 9,990 barrels of fuel oil, or 0.0292 barrel per ton of sand and gravel mined.

PLANT

Figure 3 shows a flow sheet of the plant.

The sand and gravel are loaded into drop-bottom cars and transported to the plant hopper which is 15 by 30 feet in horizontal section, is built of concrete, and has a capacity of 150 tons. A horizontal grizzly made of 60-pound rails, inverted, and spaced with 4-inch clearances, covers the entire top of the hopper. The material through the grizzly is fed by a 30-inch plate feeder to a 30-inch main conveyor belt which delivers it to the scrubber section. The plate feeder is driven from the tail pulley of the main conveyor by chain and sprocket. It has a speed of 60 r.p.m. and a throw of from 3 to 7 inches which may be regulated to suit requirements. The main conveyor belt is 185 feet between centers and is supported on an incline of 15 degrees by timber framework. This belt is driven 250 f.p.m. by a 30-hp. ball-bearing motor making 1,200 r.p.m. The belt is made of 5-ply 32-ounce duck with 1/8-inch rubber top and 1/16-inch rubber bottom. Any material that passes the grizzly but is too large for commercial use is thrown off the belt by hand into a 12-inch jaw crusher set to crush to 2 inches and operating at 600 r.p.m.

The cost of digging test pits varies with the hardness of the ground. There is little caving of the sidewalls, except below the water line. Post holes cost approximately 10 cents per foot and the test pits about $1 per foot of depth.

CHOICE OF METHOD

The dragline was chosen as the best means of excavation, for the following reasons: (1) The deposit is shallow, the average depth to the bottom of the gravel being 20 feet; hence, frequent moving of excavating equipment is required; (2) 4 to 10 feet of the deposit is below water level; (3) the railroad tracks and hauling equipment required for delivery to the washing plant could be installed and maintained with minimum difficulty; and (4) there is not enough dependable water to operate a pumping plant.

The overburden amounts to about 40 per cent of the total material handled and is easily removed without dilution of the sand and gravel below. Trees sometimes grow over the deposits. If there are many of them, or if they are large, it is necessary to cut them down before commencing excavation, but if they are small they are removed by the dragline along with the overburden. This overburden is cast back into the open pit which has just been excavated. The accompanying plan of tracks and pit operation (fig. 2) shows the method used.

STRIPPING

Stripping is ordinarily done intermittently by the same dragline that does the loading and requires about 40 per cent of its digging time. The dragline will strip off a "set up" (the area reached by the bucket without moving the machine), and then proceed to load the gravel and sand into cars before moving down the track to the next set up. The overburden is usually wasted. Occasionally, however, the railroads will use it for making fills or repairing washouts, but less than 2 per cent of the overburden is sold.

EXCAVATION

A skid-and-roller type and a walking-type dragline are used for both stripping and loading. The former is steam operated and is equipped with a 100-foot boom and a 4½-cubic yard bucket. It has a 120-hp., locomotive-type boiler with 176 two-inch tubes. Oil is used as fuel and is pumped from the locomotive tender to the dragline as required. The locomotive takes oil from the main storage tank located at the washing plant. Water is pumped to the dragline boiler from an abandoned pit in which the water has cleared by a 5-hp. gasoline engine directly connected to a 2-inch centrifugal pump. It is conveyed to the dragline by a 2-inch pipe line laid on top of the ground along the railroad track. This dragline uses 140 feet of 1-3/8-inch, 6-strand, 19-wire, Lang-lay cable with independent wire-rope center as a drag cable. The life of the cable is approximately three months. For hoisting, a 6-strand, 19-wire, regular lay, hemp-center 7/8-inch cable 440 feet long is used. This cable lasts approximately eight months under average conditions.

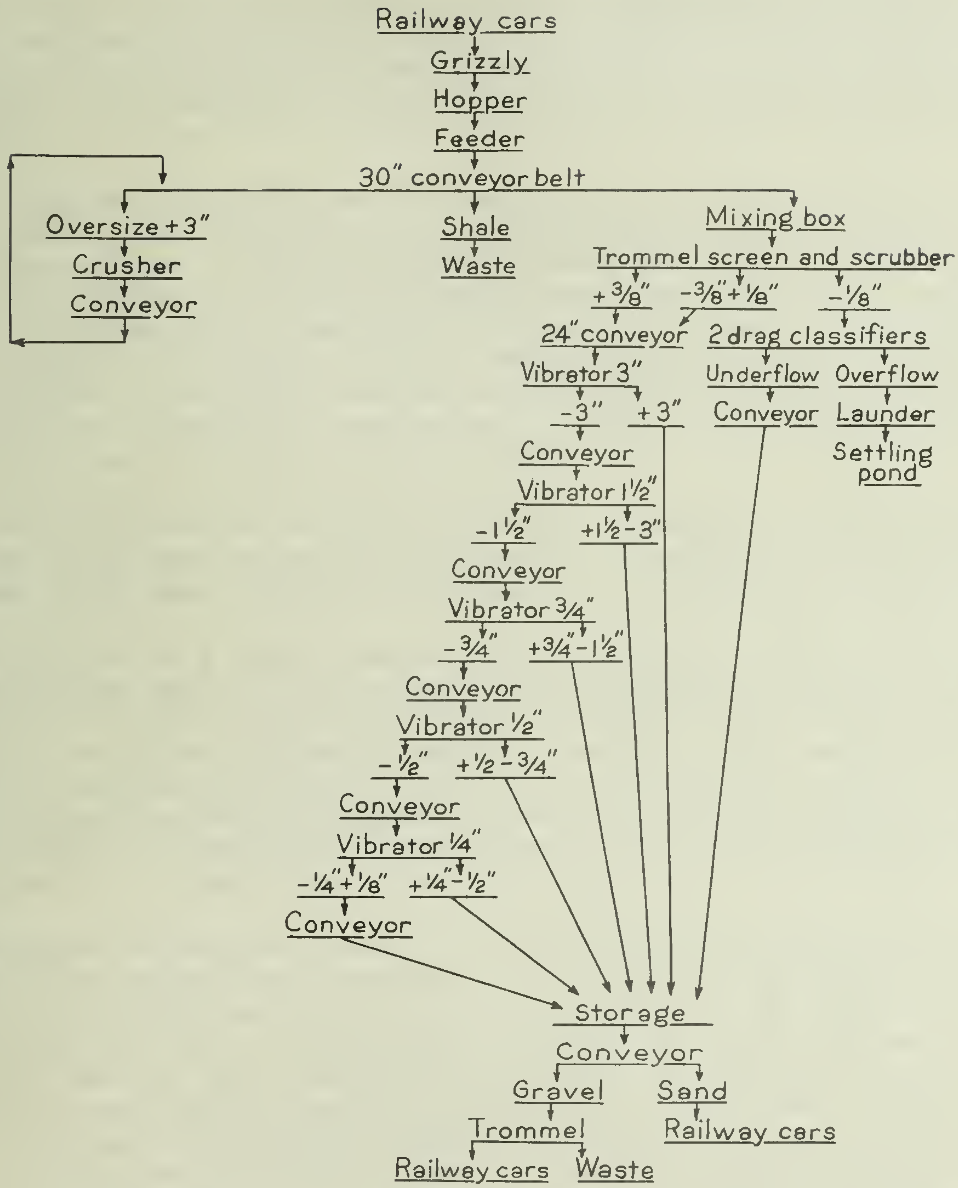

Figure 3.-Plant flow sheet

The crusher discharges onto a 16-inch inclined conveyor belt which carries the crushed material back onto the main conveyor. The crusher and 16-inch conveyor are belt driven by a 15-hp. motor making 1,200 r.p.m.

Lumps of shale that have been dug up by the draglines are picked off the belt by hand and piled on the ground until there is a carload, when they are picked up by a small clamshell and loaded into cars or trucks and hauled to waste piles. The quantity of shale or oversize is small and gives practically no trouble.

The conveyor belt delivers to a mixing box 50 feet above the bottom of the hopper. This box, 4 feet square and 3 feet deep, is made of wood and lined with steel plate. An 8-inch stream of water discharging into it mixes the material and washes it through an opening into the main washing screen. The water is supplied from a pond 75 by 100 feet dug a short distance away. This pond is about 15 feet deep, and the natural flow of ground water into it is supplemented by using the wash water over again after the silt has settled. An 8-inch centrifugal pump driven by a 50-hp. ball-bearing motor forces 1,200 gallons per minute into the mixing box.

The pipe from the main pump has a horizontal length of 400 feet and a vertical rise of 40 feet, making a total head of approximately 115 feet. The pressure gage at the pump registers an average of 40 pounds per square inch.

The main screen is a heavy-duty cylindrical scrubber, which is 6 feet in diameter and 20 feet long. It is equipped with trunnion bearings, is inclined at 3° and makes 30 r.p.m. The first 5 feet is a solid scrubber section with baffles which break loose all particles stuck together. The baffles are made of 3-inch angle iron and are set 3° out of line with the screen axis. The next 15 feet is made up of ½-inch steel plate perforated with 3/8-inch round holes. Surrounding the screen is a sand jacket of wire cloth with 1/8-inch square openings. This jacket is 6½ feet in diameter and 15 feet long.

The proportion of material passing over and through the various screens of the trommel has never been determined. Ordinarily the 3/8-inch screen lasts one year while the sand jacket has to be renewed at the end of six months. The sand and water passing the sand jacket are flumed to two sand classifiers of the drag type, constructed of wood with three railroad rails in the bottom of the tank to take the wear of the flight plates. Each box is 5½ feet wide inside and has a flat-bottomed section 10 feet long and a 15-foot section with the bottom inclined at 20°. Two lengths of No. 830 chain have attachments at every other link for carrying the flight plates, which are made of 3/8-inch steel, 5 feet long and 6 inches wide. These two chains are set on 25-foot centers. The plates are spaced at 1-foot intervals and travel 25 feet per minute. Each drag is operated by a 15-hp. ball-bearing motor making 1,800 r.p.m. and transmitting power through spur gears and a chain drive in oil.

The water flows out one end of the classifier to the settling basin, and the sand is dragged to the other end, deposited on a 24-inch conveyor belt, carried to the top of the plant, and dropped onto the first storage pile.

It is difficult to determine the tonnage handled by each of these drags, as at times the material from the pit will not contain much sand and one drag is operated without the other. At other times the flow of material is reduced while one of the drags is being repaired. If both drags are running they each handle one-half of the sand. The parts on these sand drags wear rapidly. Where the flight plates wear on the rails in the bottom of the tanks, thereby cutting a groove in the flights, they are repaired with a piece of $\frac{1}{2}$ by 6 by 6 inch steel plate, drilled with two holes through the center line and bolted so that when one side wears out they can be reversed. The drag chain is of the link, bushing, and pin type, and will last a year if the bushings are properly renewed when worn. The sprockets for this chain are made of manganese steel, having 10 renewable, reversible teeth. These sprockets will last a year.

Both sand drags deposit sand on the same conveyor. This conveyor is 172 feet between centers, is inclined at 15°, and is driven through gear connection by a 15-hp. ball-bearing motor making 1,800 r.p.m.

This conveyor and all the other belts in the plant, with the exception of the main conveyor belt, are 24 inches wide, and are made of 5-ply 32-ounce duck with 1/8-inch rubber top and 1/16-inch rubber bottom. The short belts are all driven by 5-hp. ball-bearing motors, through gear connection.

All sizes of material above 1/8 inch are discharged from the end of the cylindrical screen onto a belt conveyor and carried to the top of the plant for sizing and distribution to storage piles. The gravel is conveyed by a series of short belt conveyors, 55 to 65 feet long to a series of 5 vibrating screens, 4 by 7 feet, and inclined at approximately 40°. The screens on these vibrators will last from 6 to 15 months, depending on their mesh size. The larger screens wear out more quickly as they carry a larger load. At each successive screen the oversize is screened out and carried to storage, and the undersize goes to the next screen. Wire screens are used on these vibrators, the size of openings beginning at 3 inches and diminishing to $1\frac{1}{2}$, $\frac{3}{4}$, $\frac{1}{2}$, and $\frac{1}{4}$-inch, making sizes of gravel as follows: 3 to $1\frac{1}{2}$ inches; $1\frac{1}{2}$ to $\frac{3}{4}$ inch; $\frac{3}{4}$ to $\frac{1}{2}$ inch; $\frac{1}{2}$ to $\frac{1}{4}$ inch; and $\frac{1}{4}$ to 1/8 inch. The last size is added to the sand in proportions to meet specifications for concrete sand.

Some sizes, notably brick sand, $\frac{3}{4}$ to $\frac{1}{2}$ inch gravel, and $\frac{1}{2}$ to $\frac{1}{4}$ inch gravel are made in excess quantities, especially during the winter months. A slack-line cableway drags this surplus away from the plant in the direction indicated by arrows on the plan (fig. 4). These materials are reclaimed by reversing the cableway bucket. This storage is necessary because of the seasonal demand for the fines.

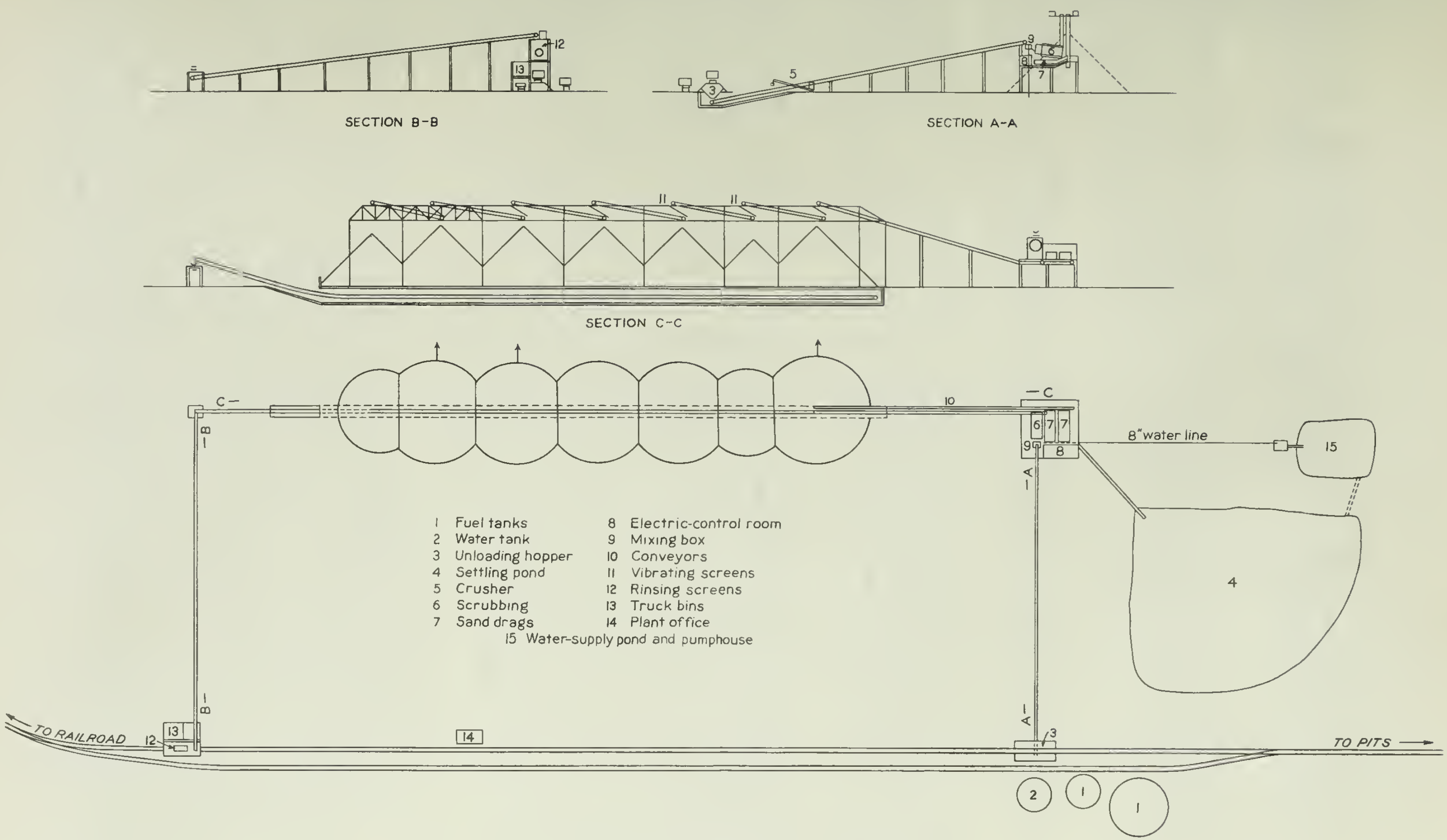

Figure 4 – Plan of Quigley plant, Hart Spur, Tex.

The cableway is equipped with a bottomless, $\frac{1}{2}$-yard bucket, pulled by a 3/8-inch plough-steel cable at a rate of 100 f.p.m. The track cable is $\frac{1}{2}$ inch. Both cabled last about one year. The cableway is powered with a 15-hp. motor, gear-connected to the drag and reverse drums.

A subsidiary company making cold-mixed asphalt buys a considerable quantity of the fines. They are also shipped to the central concrete plant of the company, where they are used in concrete mixes whenever possible. These two plants take considerable material the disposition of which would ordinarily be a problem.

The plant will easily wash and store 250 tons of material per hour. The storage will hold over 30,000 cubic yards without dragging any material away from the plant. The scraper more than doubles this storage capacity.

The storage piles are directly on the ground and there is no division wall between them. Such division is unnecessary because the gravel and sand are drawn from the center of the storage piles and the angle of repose does not permit one grade of material to flow to the gates underneath the adjoining pile, even if the latter is low.

A loading-out tunnel 7 by 7 feet inside and 400 feet long traverses the entire length of the storage piles. Under each stock pile are three or four 16 by 16 inch manually operated clamshell gates for drawing the material onto a 24-inch conveyor belt which is 475 feet between centers. This conveyor is horizontal through the length of the tunnel, then rises about 12 feet in a distance of 36 feet. The gravel travels out of the tunnel onto an inclined 24-inch conveyor belt (see section B-B, fig. 4) which carries it to the top of the loading tipple. The tipple belt is inclined about 15° and is driven by a 10-hp. ball-bearing motor, gear connected. The belts are of the same construction as the other 24-inch belts. The speed of the tunnel belt is 250 f.p.m., but that of the tipple belt is increased to 275 f.p.m., as a greater load is deposited on the horizontal tunnel belt than can be carried by the inclined tipple belt at the same speed.

The grading of the gravel is done on the tunnel belt; that is, the various sizes are put on the belt in the proportions necessary to meet specifications. Two men are required in the tunnel to proportion the material and to prevent bridging above the gates. Only a short time is required to train a man to adjust the gates to get the correct gradings. The plant foremen looks after the grading personally and checks it by making screen analyses, taking his samples from the car-loading chute. At the loading tipple the sand is dropped directly into the railroad cars through a flop gate, but gravel is diverted into a cylindrical screen 4 feet in diameter and 10 feet long, where it is sprayed by a stream of water to rinse off dust that has accumulated in the stock pile. This screen makes 30 r.p.m. and is inclined at 4°. It is a trunnion-type screen of skeleton construction and is covered with $\frac{1}{4}$-inch mesh woven-wire cloth. It is belt driven from a 10-hp.,

ball-bearing motor making 1,800 r.p.m. The various sizes of gravel carried on the belt are thoroughly mixed by the rinsing screen before going into the car. The chute for loading cars is hinged so that it can be swung from one side of the car to the other to prevent segregation of sizes. After loading, the cars are hauled to the railroad siding for shipment.

The washing plant consumed 247,900 kw.h. during 1931 or 0.0725 kw.h. per ton of sand and gravel mined.

Repairing of all equipment is done in the company's own machine shop under the direction of a master mechanic. The shop is equipped for all classes of repair work, except gear pressing and the turning of locomotive tires. Bushings and many wearing parts are made in the shop and a supply of them is kept on hand for making breakdown replacements.

RECOVERY

About 60 per cent of the material mined is sold as sand and gravel and 40 per cent is wasted. Of the 60 per cent saved, 24 per cent is sold as washed gravel, 18 per cent as washed sand, and the other 18 per cent as pit-run sand and gravel. The latter is used for railroad ballast and in making concrete base for streets where an asphalt or brick topping is used.

PAY SYSTEM

A monthly salary is paid the superintendent, foremen, and master mechanic. These men receive a 2-weeks paid vacation each year and bonuses on a profit-sharing basis, the rate being fixed by the board of directors. The other men are paid an hourly wage ranging from 30 to 75 cents depending on the class of work they do. The wage scale is based on a 10-hour day and 300 working days per year.

SAFETY METHODS AND FIRST-AID TRAINING

The foremen are the key men in the safety organization. A meeting is held once a month at which a man furnished by the underwriters of the company's liability insurance gives a talk on safety methods. He has divided the men into two groups for "no-accident competition." The competitive plan has proved most satisfactory and especially so when encouraged by company officials.

ADMINISTRATIVE ORGANIZATION

The organization chart, Figure 5, correlates each department, and shows the average number of men in each. The pit superintendent handles all production and maintenance matters, acting with the vice-president in an advisory capacity.

Table 1.- Summary of Costs

Hart Spur pit Period: January 1 to December 31, 1931.

Overburden removed .... tons 288,000 (estimated)
Sand and gravel loaded ... do 341,633
Sand and gravel washed ... do 288,279

Cost per dry ton of sand and gravel mined

	Labor	Supervision	Power	Fuel	Supplies	Total
Stripping and mining	$0.0323	$0.0123	-	$0.0034	$0.0110	$0.0590
Track maintenance	.0289	.0047	-	-	.0071	.0407
Transportation	.0050	.0010	-	.0195	.0040	.0295
Pit maintenance	.0055	-	-	-	.0260	.0315
Washing plant	.0481	.0073	$0.0207	-	.0082	.0843
Plant maintenance	.0053	-	-	-	.0157	.0210
Total direct operating cost	0.1251	0.0253	0.0207	0.0219	0.0720	0.2660
Depreciation	-	-	-	-	-	.0630
Depletion	-	-	-	-	-	.0216
Total operating cost	-	-	-	-	-	0.3506

Washing-plant operating cost based on tons washed

	Labor	Supervision	Power	Fuel	Supplies	Total
Washing plant	$0.0568	$0.0083	$0.0245	-	$0.0097	$0.0993
Plant maintenance	.0062	-	-	-	.0186	.0248
Totals	0.0630	0.0083	0.0245	-	0.[illegible]283	0.1241

Table 2.- Summary of Costs in Units of Labor, Power and Supplies

Hart Spur pit Period: January 1 to December 31, 1931.

Overburden removed .... tons 288,000 (estimated)
Sand and gravel loaded ... do 341,633
Sand and gravel washed ... do 289,279

A. Labor: (man-Hours per ton)	Stripping	Mining	Crushing	Total per ton of sand and gravel loaded
Loading	0.034	0.044	-	0.073
Transportation	-	.043	-	.043
Plant	-	-	.031	.026
Supervision	-	-	-	.009
Total	0.034	0.087	0.031	0.151

Average tons per man per shift 65.8
Labor, per cent of total operating cost 42.8

B. Power and Supplies: (per ton of sand and gravel loaded)

Dragline, distillate .... gal. 0.1602
Locomotive, fuel oil .... bbl. 0.0292
Washing plant .... kw.h. 0.0725

Fuel and power, per cent of total operating cost 12.0

Table 3.- Detailed Average Dragline Costs

Hart Spur pit — Period: January 1 to December 31, 1931.
Monighan dragline, 3-cubic yard bucket.

Overburden removed .... tons 288,000 (estimated)
Sand and gravel loaded ...... do 341,633

	Sand and gravel (341,633 tons)		Overburden (288,000 tons)		Total (629,633 tons)		Total cost per ton of sand and gravel
	Amount	Cost per ton	Amount	Cost per ton	Amount	Cost per ton	
Engineers	$1,950.00	$0.0057	$1,300.00	$0.0044	$3,250.00	$0.0051	$0.0095
Firemen	1,540.00	.0045	1,110.00	.0038	2,650.00	.0042	.0077
Foremen	2,520.00	.0073	1,680.00	.0058	4,200.00	.0061	.0123
Pitmen	1,465.00	.0043	975.00	.0034	2,440.00	.0039	.0072
Other labor	1,620.00	.0047	1,080.00	.0037	2,700.00	.0043	.0079
Total operating labor	9,095.00	0.0265	6,145.00	0.0211	15,240.00	0.0236	0.0446
Fuel	698.33	0.0022	465.55	0.0016	1,163.88	0.0018	0.0034
Grease and supplies	2,254.55	.0061	1,500.00	.0052	3,774.55	.0059	.0110
Total supplies ..	2,952.83	0.0083	1,965.55	0.0068	4,938.43	0.0077	0.0144
Shop labor	1,145.52	0.0033	760.00	0.0026	1,905.53	0.0030	0.0055
Repair supplies ...	5,373.94	.0158	3,520.00	.0122	8,893.94	.0141	.0260
Total repairs ...	6,519.46	0.0191	4,280.00	0.0148	10,799.46	0.0171	0.0315
Total dragline ..	18,567.34	0.0539	12,390.55	0.0427	30,977.89	0.0484	0.0905

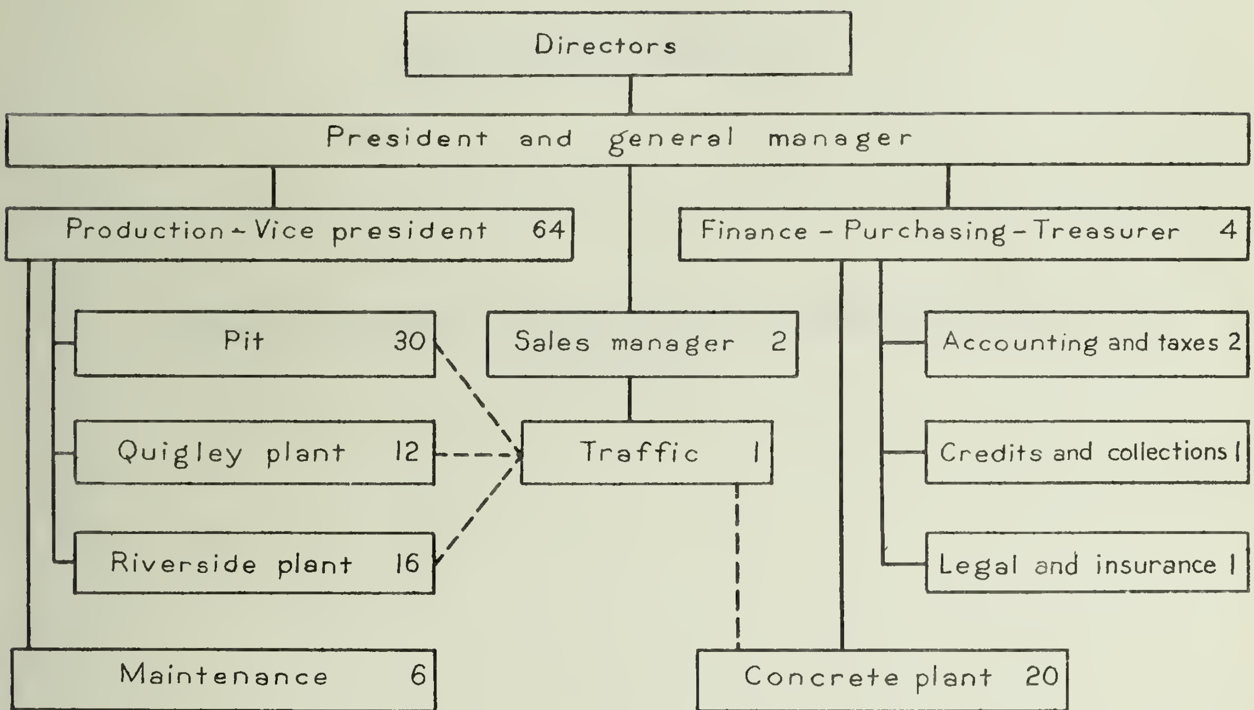

Figure 5.- Organization chart

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November, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MINING STATUTES OF THE STATE OF PENNSYLVANIA
(SUMMARIZED)

BY

J. A. Huff and V. V. Baker

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MINING STATUTES OF THE STATE OF PENNSYLVANIA (SUMMARIZED)[1]

By J. A. Huff[2] and V. V. Baker[3]

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2 Attorney, U. S. Bureau of Mines.
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INTRODUCTION

This circular is intended to present in brief form the statutory provisions of Pennsylvania regarding mining operations, in force at the end of the 1931 session of the legislature. No reference is made to general laws which affect mining only incidentally in common with other subjects, such as the workmens' compensation law. The statements have been condensed as far as practicable, and provisions not deemed of general interest, such as matters dealing only with administrative details, have been omitted.

This circular obviously is not written with such particularity as to be of great value to the legal profession, but it is hoped that it will be beneficial to the mining industry in giving this information in a relatively compact form. At the end of each paragraph the statute from which it is derived is cited to facilitate reference to the text of the law.

It is planned to publish additional circulars from time to time dealing with the mining laws of other States. Any criticisms of the form of publication followed herein, or suggestions aimed at the improvement of future circulars, will be welcomed.

CHAPTER I - GENERAL PROVISIONS

Article 1. Miscellaneous

1. Recovery of bodies of entombed workmen.- Whenever workmen are entombed or buried in a coal mine, it is the duty of the court in that county, upon petition of a relative of those entombed, to order a hearing to determine whether it is practicable to recover such workmen or their bodies; if such action appears practicable the court may mandamus the owner, lessee, or operator of the mine to proceed to work for and recover the bodies of such workmen. (1889, May 9, P.L. 154, sec. 1.)

2. Running tunnel under river to coal mine.- Whenever the power or manufacturing plant of any Pennsylvania corporation is situated near the bank of a navigable stream within the Commonwealth and such company has the right to mine coal from lands adjacent to the opposite bank of the stream, it may, upon approval of the water and power resources board, construct and operate tunnels under the stream connecting the coal lands and power plant, provided the company pays to the Commonwealth the fair market value for all coal mined in constructing the tunnels. (1919, July 8, P.L. 759, sec. 1.)

3. Endangering lives of miners or security of machinery.- It is a misdemeanor for any person knowingly to injure any safety lamp, water gage, barometer, air course, or brattice; to obstruct or throw open airways; carry lighted pipes or matches into places worked by safety lamps; disturb machinery of the hoisting engine; open a door without afterwards closing it, causing danger in the mine; enter any place of the mine against caution, or disobey any order given pursuant to this act; ride upon a loaded car in any shaft or slope, or on any plane in or around a mine; or do any other act endangering the safety of persons and the security of the mines or machinery; or for any miner having charge of a working place in a coal mine or colliery to neglect to keep the roof thereof properly propped and timbered. (1870, Mar. 3, P.L. 3, sec. 19.)

4. Miners to be paid for all coal mined, irrespective of size.- Coal miners are entitled to receive full wages accruing to them for mining of all sizes of merchantable coal, whether nut or lump coal. In adjudication of wages, 76 pounds are deemed 1 bushel, and 2,000 pounds 1 ton. This act does not purpose to prevent operators and miners from contracting for any desired method of measuring and screening the coal mined. (1883, June 1, P.L. 52, sec. 1.)

5. Inside buildings to be of incombustible material.- All buildings inside of any coal mine, including engine houses, pump houses, stables, etc., shall be constructed of incombustible material, approved in writing by the Secretary of Mines. Failure to comply with this requirement by a company or a mine superintendent is a criminal offense. (1911, June 15, P.L. 979, sec. 1-3.)

Article 2. Prohibitions concerning Employment of Females and Minors

6. Employment of minors in mines.- No minor under 16 years of age shall be employed or permitted to work in any anthracite or bituminous coal mine, or in any other mine. (1915, May 13, P.L. 286, sec. 5.)

7. Restrictions on employment of boys and females.- No woman or girl and no boy under 16 years of age shall be employed in any mine, nor shall a woman or girl or boy under 14 years of age be employed in or about the outside structures or workings of a colliery. This prohibition, however, shall not affect the employment of a boy or female, of suitable age, in an office or in the performance of clerical work at a colliery. (1891, June 2, P.L. 176, amended 1903, May 13, P.L. 359.)

8. Employment in coal breakers or washeries.- A minor under 14 years of age may not be employed or permitted to work in, about, or for any coal breaker or washery, or in or about the outside workings of any coal mine. (1911, June 15, P.L. 983, sec. 1.)

9. Employment of minors and females generally; boys under 18.- No woman or girl and no boy under 14 years of age shall be employed in or about any mine. No boy under 18 years shall be permitted to mine or load coal in any working place unless in company with an experienced person over 18 years of age. No boy under 16 years of age shall be employed in or about any mine unless there is on file at the mine an employment certificate issued by the superintendent of public schools, the secretary of the school board, a principal of a parochial school, or by such superintendent's, secretary's, or principal's duly appointed deputy or assistant, reciting the age of the boy. This section does not prohibit the employment of a girl between 14 and 16 years of age in the office of a mine, provided a like certificate is on file in that office. (1911, June 9, P.L. 756, Art. XVIII, sec. 1.)

10. Restrictions on hours of employment.- No minor under 16 years of age shall be employed in or about any establishment named in paragraph 8, ante, more than 10 hours a day, except when a different apportionment is made for the sole purpose of shortening a workday for one day in the week; nor shall less than 30 minutes be allowed for the midday meal; and in no case shall the hours of labor exceed 58 in any one week. No minor under 16 years shall be permitted to work between the hours of 9 p.m. and 6 a.m. (1911, June 1, P.L. 537, sec. 1.)

11. Proof of age.- When an employer is in doubt as to the age of any boy applying for employment in or about a mine or colliery, he shall demand and receive proof of his lawful employment age, by certificate from the parent or guardian before the boy shall be employed. (1891, June 2, P.L. 176, Art. IX, sec. 2.)

12. Penalty for violations; false certificates.- Violations of the above provisions and false swearing as to the age of any boy or girl are misdemeanors. (1911, June 9, P.L. 756, Art XVIII, sec. 2; 1891, June 2, P.L. 176, Art. IX, sec. 3.)

Article 3. Collection of Statistics

14. Statistics and information.- It is made the power and duty of the department of internal affairs to: (a) Collect, compile, prepare, and publish statistics and uniform data and information relating to labor, coal mining, oil and gas production, and other business interests of the State; (b) Transmit to the department of property and supplies for publication such reports of the statistics and information collected and compiled as shall be necessary and render such information available for the use and benefit of the public. (1929, Apr. 9, P.L. 177, Art. XII, sec. 1205.)

15. Reports of coal and coke transported.- The several railroad and canal companies passing through any of the coal regions of the State, and all slack-water navigation companies conveying coal or coke, shall annually report to the auditor general, as soon after January 1 in each year or the close of the companies' fiscal year as the information can be procured, the quantity of coal of each kind and of coke, received for transportation at each station and coal shipping point, distinguishing the quantities received direct from the mine from that received from other railroads or canals in such manner that the amount of the production of coal on the line of the railroad or canal may be correctly ascertained. (1871, May 9, P.L. 261, sec. 1.)

16. Reports by railroad companies of coal purchased or mined for own use.- Railroad companies shall report the quantity of coal purchased or mined for their own use in this State which was produced along the line of the railroad (stating at what place or places the coal was mined) and which was not included in the reports provided for in the preceding paragraph. (Id., sec. 2.)

17. Furnishing of information on request of auditor general; report of mine accidents.- All coal mining companies and State and county officers shall furnish to the auditor general, in answer to his letters or circulars, all information in their possession regarding the quantity of coal mined that is sent to market direct by any navigable river, or used by any rolling mill, blast furnace, salt works, or otherwise, and which is not transported on any railroad, canal, or by any slack-water navigation company, and shall inform him when and of whom correct information as to the coal production of any such locality can be procured; and further, inform him of all accidents in mines in counties where there is no mine inspector, and how the accidents were caused. (Id., sec. 3.)

18. Reports by auditor general.- The auditor general shall, on receiving the reports, and such other authentic information as he shall collect, collate the reports and information, and make a report, giving the results only in tabular form, showing in a perspicuous form the quantity of coal mined during each year in each county and in each important coal-producing region, separating the several kinds of coal into anthracite, semibituminous, bituminous, and splint or block coal suitable for smelting iron, giving also from time to time the statistics of each region, from the beginning of its coal trade, so far as it can be ascertained; he shall also specially report the number of accidents resulting in death or injury in coal mines in those counties where there is no mine inspector, classifying them according to the cause thereof, whether occasioned by fire, explosions, falls of roof or coal in shafts or slopes, or by other causes underground or at the surface. (Id., sec. 4.)

19. Reports on other mineral productions.- The auditor general shall also, in the same manner, collect statistics, collate, classify, and report, at the same time, the quantities of petroleum, salt, iron ore, zinc, and other mineral productions of the Commonwealth; also the pig iron and merchant or wrought iron manufactured in the Commonwealth. (Id., sec. 5.)

20. Printing and distribution of report.- Eight thousand copies of the report of the auditor general shall be published for distribution, annually, with the title "Mineral Statistics of Pennsylvania." One copy shall be sent by the auditor general to each person who shall have furnished him with information, and the balance shall be delivered to the legislature for distribution. (Id., sec. 6.)

21. Duties of auditor general transferred to secretary of internal affairs.- Hereafter the secretary of internal affairs, in lieu of the auditor general, shall send out the blanks required by the act of May 9, 1871 (Paragraphs 15-20, ante), and the secretary shall perform all the duties enjoined in the act in regard to collecting, compiling, and publishing a report of the same number of copies ordered to be published by the auditor general. (1874, May 15, P.L. 193, sec. 3.)

Article 4. Department of Mines

22. Department of mines established.- There is hereby established a department to be known as the department of mines, which shall be charged with the supervision of the execution of the mining laws of this Commonwealth and the care and publication of the annual reports of the inspectors of coal mines and any and all other mines that may come under the provisions of the mining laws of this Commonwealth. (1903, Apr. 14, P.L. 180, sec. 1.)

23. Secretary of mines.- The governor, with the advice and consent of the senate, appoints the secretary of mines, who is the head of the department of mines and serves for four years. The secretary receives a salary of $10,000 per annum. (1929, Apr. 9, P.L. 177, Art. II, secs. 206-209.)

24. Qualifications of secretary.- The secretary of mines[4] shall be a competent person having at least 10 years of practical experience as a miner and the qualifications of the present mine inspectors. (1903, Apr. 14, P.L. 180, sec. 3.)

25. Powers and duties of department in general.- The department of mines shall have the power and its duty shall be:

(a) To see that the mining laws are faithfully executed and to cause mine inspectors to enter, inspect, and examine any mine or colliery in the Commonwealth and the works and machinery connected therewith.

(b) To give such aid and instruction to the mine inspectors as may be calculated to protect the health and promote the safety of all persons employed in and about mines.

(c) To carry out such examinations and investigations as may be necessary to enable the department to make recommendations upon matters pertaining to the general welfare of coal miners and others connected with mining and the interests of mine owners and operators.

(d) Through a separate bureau to take such steps as the department may deem advisable to promote the welfare of the mining and mineral interests of the Commonwealth, and the use of the mineral products of Pennsylvania. The exercise of this function shall not in any way interfere with the safety work of the department. (1929, Apr. 9, P.L. 177, Art. XIX, sec. 1902; amended, 1931, June 1, P.L. 350.)

26. Power to inspect mines; suspension and removal of inspectors.- The secretary of mines shall devote the whole of his time to the duties of his office and see that the mining laws are faithfully executed. He is invested with the same power and authority as the mine inspectors to inspect any mine or colliery in the State. The secretary may suspend any mine inspector for neglect of duty, but such inspector may appeal to the governor, who is empowered to approve the suspension or restore the inspector to duty. Upon receiving a petition signed by 10 or more miners or three or more operators setting forth that a mine inspector is neglectful, or physically unable to perform his duties, or guilty of malfeasance, the secretary of mines shall investigate the matter, and if he finds the charges well founded shall institute court proceedings for the removal of the inspector. If the charges are proved, the court shall certify the same to the governor who shall certify the office vacant and proceed to fill the same.[5] (1903, Apr. 14, P.L. 180, sec. 4.)

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4 Under the act of April 14, 1903, cited to the text, the head of the department of mines is referred to as the chief of the department of mines. Later enactments, however, have changed this designation to secretary of mines, and wherever the former designation appears it has been changed in this compilation to read "secretary of mines."

5 See also paragraphs of this paper numbered 39 and 203.

27. Reports of mine inspectors; investigations.- The secretary of mines shall preserve in his office the annual reports of the mine inspectors and transmit a synopsis of them and other work of the department as may be of public interest, to the governor on or before the 15th day of March in each year. The mine inspectors shall deliver their annual reports to the secretary on or before the 20th day of February in each year. The mine inspectors shall furnish the secretary of mines with monthly reports and such other information as he may require or deem necessary in the discharge of his official duties. The secretary shall establish a uniform style and size of blanks for the various reports of the mine inspectors.

The secretary of mines is authorized to make such examinations and investigations as may enable him to report on the various systems of mining practiced in the State, method of mining ventilation and machinery employed, circumstances and responsibilities of mine accidents, and such other matters as may pertain to the general welfare of miners and others connected with the mining industry. (Id., sec. 5.)

28. Duplicates of manuscripts to be filed in department.- The board for examination of applicants for mine inspectors in the anthracite and bituminous coal mines, the board for examination of applicants for mine foremen and assistant mine foremen in the anthracite mines, the board for the examination of applicants for first and second grade certificates in the bituminous mines, and the miners' examining board, shall send to the secretary of mines duplicates of the manuscripts and all other papers of applicants, together with the tally sheets and the solution of each question as given by the examining board, which shall be filed in the department as public documents. (Id., sec. 6.)

29. Record of inspections, etc.- The secretary of mines shall keep in the department a record of all inspections, examinations, and work done under his administration, and copies of all official communications. All instruments, plans, books, and records pertaining to the office shall be the property of the State. (Id., sec. 8.)

30. Secretary accountable to governor; certain persons ineligible for office.- The secretary of mines is at all times accountable to the governor for the faithful discharge of his duties, and the rules and regulations pertaining to the department are subject to the approval of the governor. No person acting as a manager, viewer, or agent of any mine or colliery shall at the same time serve as secretary of mines. (Id., secs. 10, 11.)

CHAPTER II - ANTHRACITE COAL MINES

31. Introduction; mines affected by this chapter.- This chapter is taken mainly from the act of June 2, 1891 (P.L. 176), and its amendments, regulating anthracite coal mining in Pennsylvania. By its terms this act applies only to anthracite coal mines or collieries in the State employing more than 10 persons. Paragraphs numbered 57-65, 70-72, 162-168, and 178-188, however, are taken from other statutes and, except where otherwise noted, those paragraphs apply to all anthracite mines in the State.[6]

32. Id.; definitions.- The act of June 2, 1891, as amended by the act of June 1, 1915 (P.L. 712), provides that, unless the context otherwise requires:

"Coal mine or colliery" includes every operation and work both under and above ground, used for mining and preparing coal.

"Workings" includes all the excavated parts of a mine; those abandoned as well as places actually at work.

"Mine" includes all underground workings and excavations; shafts, tunnels and other ways and openings; also all such shafts, etc., in course of being driven; and all roads, appliances, machinery, and material connected with the same below the surface.

"Shaft" means a vertical opening through the strata used for ventilation, drainage, or hoisting men or material.

"Slope" means any inclined way or opening used for the same purpose as a shaft.

"Breaker" means the structure containing the machinery used for the preparation of coal.

"Owner" and "operator" means any person or corporation who is the immediate proprietor, lessee, or occupier of any coal mine or colliery; it does not include one who merely receives a rent or royalty, or is merely proprietor subject to any lease, grant, or license for the operation thereof, or is merely the owner of the soil and not interested in the minerals of the mine. But any "contractor" for the working of a mine or colliery shall be subject to this act as an operator or owner.

"Superintendent" means the person having, on behalf of the owner, general supervision of one or more mines or collieries.

"Miner" means the person who cuts or blasts coal or rock at the face of a gangway, airway, breast, pillar, or other working place; also any person engaged at general work in a mine and qualified to do the work of a miner.

The definitions given herein apply only to these terms when used in the act of 1891 and its amendments; but do not, at least by legislative declaration, apply to these terms when used in independent enactments treated in the paragraphs listed in paragraph 31.

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6 See also paragraph 190 as to mines outside the anthracite district.

Article 1. Inspectors and Inspection Districts

33. Inspection districts.- The anthracite region is divided into eight inspection districts, as follows: First district, County of Luzerne; second, Lackawanna; third, Carbon; fourth, Schuylkill; fifth, Northumberland; sixth, Columbia; seventh, Dauphin; and eighth district, Counties of Susquehanna, Wayne, and Sullivan. (1911, May 5, P.L. 120, sec. 1.)

34. Mine inspectors examining board; personnel, etc.- The anthracite mine inspectors examining board shall consist of the superintendent of public instruction, ex officio; the secretary of mines who is chairman; two mining engineers, having had at least five years of experience in the anthracite mines of Pennsylvania; and three members who shall be coal miners in actual practice having had at least five years of practical experience in the anthracite mines of the State. The last named five members are appointed by the Governor. The members must be at least 30 years of age. The board may elect a secretary who need not be a member. Each member receives $15 per day while actually engaged in performance of work. (1921, May 17, P.L. 831, sec. 1; 1929, Apr. 9, P.L. 177, Art. IV, sec. 424.)

35. Organization of board; oath.- The examining board shall meet in Wilkes Barre on the second Tuesday in September following its appointment to prepare questions and formulate rules governing the examination, provided there be a vacancy in the office of inspector. After the board is organized each member shall take an oath to perform his duties faithfully and impartially. (1921, May 17, P.L. 831, sec. 2.)

36. Examination of candidates for office of inspector; qualifications.- The anthracite mine inspectors' examining board shall meet in Wilkes Barre on the fourth Tuesday of September to examine applicants for the office of inspector. Candidates must be citizens of Pennsylvania, residents of the anthracite region, of temperate habits, good repute, and personal integrity, in good physical condition, and not under 30 or over 50 years of age. They must have a comprehensive knowledge of the different systems of working and ventilating coal mines; have had at least 10 years of practical experience in anthracite mines of Pennsylvania, 5 of which must be as coal miners; have had practical experience with explosive gas, dangerous and noxious gases found in coal mines, and have a general knowledge of mining and machinery and the chemistry of gases found in coal mines. They must be conversant with the work of the first aid and rescue corps, with the science and use of electricity as applied to coal mines, be able to understand and read the workings of any mine as shown on maps, and to make a cross section of any mine from said maps. They must give evidence of such theoretical and practical knowledge respecting mining as will satisfy the board of their capability for the office of inspector. The examination is in writing, and 90 per cent is a successful average. Examinations may not exceed 20 days in duration. The examining board certifies to the governor and secretary of mines the names and grades of successful applicants, each of whom receives a certificate of qualification. Such certificate remains in force for four years only, unless the holder has served a full term as mine inspector in the anthracite mines, in which case the certificate becomes permanent. (1921, May 17, P.L. 831; Appropriation Acts 1917, July 16, p. 64, sec. 2.)

37. Appointment of inspectors; salaries; bond.- When a vacancy occurs in office of anthracite mine inspector the governor commissions for a term of four years, from the names certified to him, the person having the highest percentage. Where a vacancy occurs before expiration of a full term, the appointment is for the unexpired term. When the list of successful applicants is exhausted, the governor shall cause the examining board to hold a special examination, In case of incapacity or of an inspector's being granted leave of absence for 30 days or more, on request of the secretary of mines the governor shall appoint a person on the eligible list to act in the inspectors stead.

Inspectors receive a salary of $4,800 per year and are entitled to incur traveling and other necessary expenses. Before entering upon their duties they must give bond in sum of $5,000 and take an oath to discharge their duties with impartiality and fidelity. (1921, May 17, P.L. 831.)

38. Inspectors not to be interested in mines.- Inspectors may not act as manager, agent, or mining engineer for any coal company, or be interested in the operation of any anthracite coal mines in Pennsylvania. (1921, May 17, P.L. 831, sec. 14.)

39. Removal of inspectors; filling vacancy.- Upon petition of 15 reputable miners or mine operators, with affidavit of one or more attached, or upon petition of the secretary of mines, setting forth that any inspector of mines is holding office illegally, or is neglectful, incompetent, or guilty of malfeasance, the court of common pleas in the county in which the inspector is a resident, shall issue a citation to said inspector to appear, at not less than 10 days notice, and investigate the charges. If they are sustained the court shall declare the office vacant and so certify to the governor, who shall fill the vacancy, in which event the costs are imposed on the inspector. If the charges are not sustained but the court believes there was sufficient ground for them, the costs are placed on the county. (1921, May 17, P.L. 831, sec. 15.)

40. Residence of inspectors; duties in general.- Each inspector must reside in the district for which appointed and devote all his time to the duties of his office. He shall examine all collieries in his district at least once every three months and oftener if necessary, see that necessary precaution is taken to secure safety of workmen, and shall visit each working face and see that it is supplied with a sufficient quantity of air current. Every three months he shall make a report of the condition of each working face in each colliery, on a form furnished by the secretary of mines, designating the gangway in which the working is situated, the breast number, and their condition as good, fair, or bad. The report shall be placed in a conspicuous place at each mine opening.

The inspector shall certify in the report that the employees are hoisted to the surface or given access thereto according to law. He shall attend every inquest held by the coroner upon the bodies of persons killed in or about

the collieries in his district; he shall visit the scene of the accident for making an examination wherever loss of life or serious personal injury occurs, and at the close of every year shall make a report of his proceedings to the secretary of mines, enumerating all accidents in and about the collieries in his district, the condition of the workings of the said mines with regard to safety of workmen therein and the ventilation thereof, and the results generally shall be fully set forth. (1905, May 3, P.L. 363, sec. 1.)

41. Powers of inspector; duty of mine owner.- The mine inspector is empowered, and it is his duty, to inspect any mine or colliery in his district at all reasonable times, but not so as to obstruct its operation, and he may take fellow-inspectors with him. He has the right, and it is his duty, to make inquiry into the condition of such mines, their workings, etc., and into all matters relating to, as well as to make suggestions providing for, the health and safety of employees, and especially to inquire whether the provisions of the act of 1891 have been complied with. The owner, operator or superintendent of such mine or colliery is required to furnish the means necessary for such inspection. The inspector shall make a record of the visit, noting the time and material circumstances. (1901, June 8, P.L. 535, sec. 1.)

42. Records to be transferred to successor.- The maps, plans, and records of the mines shall be preserved by the inspector and transferred to his successor in office. (1901, June 8, P.L. 535, sec. 1.)

43. Inspection of collieries in counties not named in paragraph 33.- The secretary of mines shall direct one or more inspectors to inspect anthracite mines and collieries in counties not mentioned in paragraph 33 of this paper, and the inspectors shall make and include in their return a report of the inspection. (1901, June 8, P.L. 535, sec. 1.)

Article 2. Surveys, Maps, and Plans

44. Accurate maps to be made and copies furnished to inspectors.- The owner, operator, or superintendent of every coal mine or colliery shall make an accurate map or plan of the workings of such mine, on a scale of 100 feet to the inch, which shall exhibit the workings or excavations in each seam of coal and connecting tunnels and passages. It shall state the general inclination of the strata, with material deflections, and the tidal elevations of the bottom of each shaft, slope, tunnel, and gangway, and of any other point where such elevation is deemed necessary by the inspector. It shall show the number of the last survey station and date of each survey on the gangways or the most advanced workings, the boundary lines of the lands of the mine and proximity of the workings thereto; and if a mine contains dammed-up water, the true location of the dam shall be shown, together with the elevation, inclination of strata, and area of workings containing water. Whenever any workings are approaching the workings where such dam or water is situated, the owner, operator or superintendent shall notify the inspector without delay. The operator shall deposit with the inspector a true copy of the map or plan

showing the workings of each seam, if so desired by the inspector, on a separate sheet of tracing muslin. One copy shall be kept at the colliery. Every six months there shall be placed on the inspector's map the plan of the extensions made during the preceding six months. (1891, June 2, P.L. 176, Art. III, sec. 1, 2.)

45. Maps of abandoned mines.- When a mine is worked out preparatory to abandonment, or when a lift thereof is about to be abandoned, the operator shall have the maps extended to include all excavations, as far as practicable. Portions worked to the boundary lines of adjoining properties, or any part which it is intended to be allowed to fill with water, must be surveyed in duplicate, and such surveys must practically agree, and certified copies of same be filed with the inspector. (Id., sec. 3.)

46. Failure to furnish map; correction of inaccuracies.- If the operator fails for three months to furnish such map or to make extension thereon, the inspector may cause an accurate map to be made at the expense of the owner. If the inspector finds or has reason to believe that any map furnished is materially inaccurate, it is his duty to apply to the court of common pleas for an order to have an accurate map prepared. If such survey shall prove that the map was inaccurate the owner shall be liable for the expense incurred, otherwise the Commonwealth shall be liable. (Id., sec. 4-6.)

47. Maps and plans property of State; kept open for inspection.- Maps or plans of mines placed in the custody of the inspector shall be the property of the Commonwealth. Copies may not be made without consent of the operator of the mine desired to be copied. The map of any particular colliery may be inspected, in the presence of the inspector, by any miner of that colliery whenever he shall have cause to fear that his working place is becoming dangerous because of proximity to other workings which may be supposed to contain water or dangerous gases, and shall be open to examination of any interested citizen during business hours. (Id., sec. 8, 9.)

48. Duties of owners of adjoining properties.- Owners of adjoining coal properties shall leave a pillar of coal in each seam or vein worked by them along adjoining property, of such width that taken with the pillar to be left by the adjoining owner, will be a sufficient barrier for the safety of employees of either mine in case the other is abandoned and allowed to fill with water. Such width shall be determined by the engineers of the adjoining property owners with the inspector, and the surveys of the face of the workings along such pillar shall be made in duplicate and must practically agree. A certified copy of such surveys must be filed with the owners of adjoining properties and with the inspector of the district. (Id., sec. 10.)

Article 3. Reports and Notices by Owner or Operator

49. Annual reports to be made.- The owner, operator, or superintendent of every mine shall send to the district inspector an annual report specifying the name and post-office address of the operator and officials of the mine; the quantity of coal mined; the amount of powder or other explosives consumed; and the number of persons employed above and below ground, classified. The report shall be in such form as prescribed by the inspector, and blank forms for the purpose are to be furnished by the State. (1891, June 2, P.L. 176, Art. XIV, sec. 3.)

50. Immediate notice in certain cases.- The owner, operator, or superintendent shall without delay give notice to the inspector: Where any working is commenced for opening a new slope or mine; where any mine is abandoned or workings thereof discontinued; where working of any mine is recommenced after discontinuance exceeding three months; where any new coal breaker is completed and work commenced therein; where the pillars of a mine are to be removed or robbed; where a squeeze or crush or any other cause or change may seem to affect the safety of employees; or where fire occurs or a dangerous body of gas is found in a mine. (Id., sec. 2.)

51. Reports of accidents.- See paragraphs 155 and 156 of this paper.

Article 4. Mine Foremen and Fire Bosses

52. Qualifications of foremen and assistant foremen.- It is unlawful for any person to act as mine foreman or assistant mine foreman of any coal mine or colliery unless registered as a holder of a certificate of qualification or service, or unless in the judgment of the employer the person is possessed of qualifications which make him equally competent to act in such position. (1915, June 1, P.L. 712, sec. 2.)

53. Board of examiners; certificates of qualification.- For examination of candidates for such certificates, a board of examiners shall be appointed in each inspection district. Such board shall consist of the district inspector, ex officio, two practical miners, and one mine owner, operator, or superintendent, and shall act for one year from the date of appointment. The board may conduct such examinations as in their judgment seems proper, and shall report their action to the secretary of mines and certify to the qualification of successful candidates. No examination may exceed 10 days in duration. Certificates of qualification to mine foremen and assistant mine foremen in the anthracite mines shall be granted by the secretary of mines to each successful applicant. (1891, June 2, P.L. 176, Art. VIII, sec. 3; Appropriation Acts 1917, July 16, p. 64, sec. 2; 1923, May 28, P.L. 456, sec. 1.)

54. Foremen required; penalty for operation without.- No mine shall be operated longer than 30 days without supervision of a mine foreman. For violations the owner, operator, or superintendent shall be subjected to a penalty of $20 per day for each day over the 30 days. (1915, June 1, P.L. 712, sec. 3.)

55. Mine foreman, fire boss, etc., agents of owners.- The mine foreman, assistant mine foreman, fire boss, and any person placed in charge of the works, or any part thereof, shall be the agent of the owners and operators, who shall employ them and discharge them at will. (Id., sec. 6.)

56. Qualifications of fire bosses.- No person shall be permitted to act as fire boss in any coal mine or colliery unless he has had five years of practical experience as a miner, three of which have been in mines wherein noxious and explosive gases are evolved; and the fire boss shall certify to the same before entering upon his duties before an alderman, justice of the peace or other person authorized to administer oaths. A copy of the deposition shall be filed with the district inspector of mines. (1891, June 2, P.L. 176, Art. VIII, sec. 9.)

Article 5. Miners' Examining Boards; Qualifications of Miners

57. Miners' examining boards.- There shall be established in each of the eight inspection districts in the anthracite coal region a miners' examining board to consist of nine miners appointed as the boards to examine mine inspectors are now appointed from the most skillful miners actually engaged in mining in their respective district, who must have had five years of practical experience in that district. The said persons serve for two years and receive compensation of $3 per day while actually engaged in this service and all legitimate expenses incurred in attending the meetings. (1897, July 15, P.L. 287, sec. 2.)

58. Organization of boards.- The miners' examining boards shall organize by electing a president and secretary and dividing themselves into three sub-committees. Each of the committees shall have all powers conferred upon the board. Members of the board must take an oath or affirmation faithfully and impartially to discharge the duties of their office. Vacancies on the board are filled in the same manner as original appointments are made. (Id., sec. 2.)

59. Meetings of committees; records of proceedings.- Each board shall designate a place within its district for the meeting of the committees thereof, of which notice shall be given by advertisement in two or more newspapers. Meetings shall not be held in a building where intoxicating liquors are sold. An accurate record of its proceedings shall be kept which shall show a correct detailed account of the examination of each applicant, and at its meetings the board shall keep said record open for public inspection. (Id., secs. 3, 5.)

60. Registration of miners.- Each of said committees shall open at the place of meeting a book in which shall be registered the name and address of each person qualified under this act to be employed as a miner in an anthracite coal mine. It is the duty of all persons employed as miners to be properly registered, and in case of removal to another district it is his duty to be registered in the new district. Application for registration only may be mailed to the board after being properly attested. The form of application is subject to regulations prescribed by the boards, but applicants shall not be put to unnecessary expense. (Id., sec. 3.)

61. Fee for examination and registration.- Applicants for examination and registration must pay a fee of $1, and a fee of 25 cents is charged for registering a person who has been examined and registered by another board. Money so received is held by said boards and applied to expenses and salaries. The board shall report annually to the court of common pleas of their respective counties and the department of mines all moneys received and disbursed, together with the number of miners examined and registered and the number failing to pass the examination. (Id., sec. 4.)

62. Meetings of boards; certificates of competency.- Each board (or committee) shall hold a public meeting once every month, which may be continued to cover three days in succession, and shall examine under oath all persons desiring to be employed as miners in their respective districts and shall grant qualified persons certificates of competency, entitling them to be employed as and do the work of miners as may be expressed in the certificate. Such certificate is good evidence of registration and the holder is entitled to be registered without examination in any other of the anthracite districts. (Id., sec. 5.)

63. Issuance of certificates; qualifications of applicants.- Certificates shall be issued only at meetings of the board and must be then and there signed in person by at least three members. Persons applying for a certificate must produce satisfactory evidence of having had at least two years of practical experience as a miner or mine laborer in the mines of Pennsylvania. Applicants shall not be deemed competent unless they appear in person and answer intelligently and correctly at least 12 questions in the English language pertaining to the requirements of a practical miner, and shall be properly identified under oath as a mine laborer by at least one miner holding a certificate. Certificates are not transferable and any transfer of a certificate is a violation of the act. (1897, July 15, P.L. 287, sec. 5.)

64. Board to investigate complaints and prosecute offenders.- It is the duty of the miners' examining boards to investigate complaints and charges of violations of the act and to prosecute persons so offending. Upon the failure of the board so to do they are subject to prosecution by the district attorney. Any resident of the State may also prosecute persons violating the act and has power to employ private counsel. Upon conviction of a member of the board for a violation of the act, in addition to the penalties provided, his office shall be declared vacant. For the purposes of this act members of the several boards have power to administer oaths. (Id., secs. 9, 10.)

65. Employment of miners without certificate of competency prohibited; penalty.- No person shall engage or be employed in the anthracite coal region as a miner in any anthracite coal mine without having obtained a certificate of competency and qualification from the miners' examining board of the proper district and having been duly registered. Violations of this act are punishable by fine and imprisonment. (Id., secs. 1, 6.)

Article 6. Qualifications, Duties, and Hours of Labor of Particular Employees

66. Hoisting engineers.- An engineer in charge of an engine whereby persons are hoisted or lowered in any mine shall be a sober and competent person not less than 21 years of age. Every engineer shall work his engine slowly and with great care when any person is being lowered or hoisted in a shaft or slope, and no one shall interfere with or intimidate him while in the discharge of his duties. He shall be in constant attendance for hoisting and lowering persons while any person is below ground, and shall allow no one except such as may be deputed by the owner, operator, or superintendent to meddle with the engine or machinery under his charge. No person engaged as a hoisting engineer at or about the anthracite coal mines, part of whose duties it is to lower and hoist persons and coal into and from the mines, shall work longer than eight hours out of each day of 24 hours; violations of this provision are a misdemeanor punishable by fine of not less than $25 nor more than $100. (1891, June 2, P.L. 176, Art. XII, rules 18-20; 1911, Apr. 29, P.L. 102, secs. 1, 2.)

67. Outside foreman.- The owner, operator, or superintendent of a colliery shall place a competent person, to be called "outside foreman" in charge of the breaker and outside work of such colliery, who shall direct and see that the provisions of this act are complied with in respect to the breaker, outside machinery, ropes, cages, and all other things pertaining to the outside work, unless otherwise provided for in this act. (1891, June 2, P.L. 176, Art. XII, rule 22.)

68. Duties of fireman in charge of boilers.- Every fireman in charge of a steam boiler shall constantly watch it to see that the pressure does not exceed the limit allowed by the outside foreman or superintendent. He shall frequently try the safety valve and shall not increase the weight thereon. He shall maintain a proper depth of water in each boiler, and should anything prevent this, shall report the fact without delay to the foreman in charge, and take other necessary action for protection of life and property. (Id., rule 39.)

69. Headman and footman.- At every shaft or slope in which provision is made in this act for lowering and hoisting persons, a headman and footman shall be designated by the superintendent or foreman to be at their proper places from the time persons begin to descend until all the persons who may be at the bottom when quitting work shall be hoisted. They shall personally attend to the signals and see that the provisions of this act, in respect to lowering and hoisting persons, shall be complied with. (Id., rule 40.)

Article 7. Weighing and Record of Coal Mined

70. Weighing of coal for ascertaining compensation of miners.- All persons and corporations engaged in mining anthracite coal in this Commonwealth shall erect at each of their coal mines standard scales for weighing the coal mined therein; and each miner's coal shall be separately and accurately weighed thereon before the coal is dumped and taken from the car. A separate and accurate account shall be kept by all persons and corporations of the coal mined by each miner. Miners shall have the right to employ at their own expense and keep a weighmaster at each scale to inspect the scales and keep an account of the coal mined by each miner; and the miners shall be paid at the rate of so much per pound for coal mined by them. This act applies only to mines in which the coal mined has heretofore been paid for by the car. Failure to comply with this act is subject to penalty of $100 for each day of such failure. The act shall not apply to persons or corporations that shall by contract agree with their miners otherwise than as provided in this act for the compensation of mining the coal.[7] (1875, March 30, P.L. 38, sec. 1.)

71. Removal of ticket showing who has loaded coal car.- Any person who shall wilfully, from any loaded coal car in or about any mines, remove or destroy any ticket or other device used to identify the persons to whom credit or pay shall be due for mining or loading of coal in the car, for the purpose of defrauding such persons in any manner, shall be guilty of a misdemeanor, punishable by fine not exceeding $100 or imprisonment not exceeding one year, or both. The jury trying the case may infer such intent from the fact of taking, removing, etc. (1907, May 28, P.L. 270, sec. 1.)

72. Record of coal mined.- At anthracite coal mines where coal is mined and paid for by the car, a record of all cars of coal mined shall be kept at the miners' chutes, or at the most convenient of the chutes, where the coal is loaded, which record shall be the final basis in computing the miners' earnings, without deduction for slate or refuse that may be loaded on the cars in the usual course of mining and loading coal, and which record shall be open for inspection of miners. Violations of this paragraph constitute a misdemeanor punishable by fine of not less than $50 nor more than $100. This act shall not affect existing contracts nor prevent the making of contracts between an operator and miners as to the method of recording cars mined and of deducting for refuse therein. (1913, July 25, P.L. 1038.)

Article 8. General Rules for Securing Health and Safety and Protection of Property

73. Precautions required; supervision.- The owner, operator and superintendent of a mine shall use every precaution to insure the safety of workmen, whether provided for in this act or not, and shall have supervision, direction, and control of the mine foreman and all other mine employees. (1915, June 1, P.L. 712, sec. 4, amending, 1891, June 2, P.L. 176, Art. XII, rule 1.)

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7 See also paragraph 4 of this paper.

74. Assistant mine foremen.- Whenever a mine foreman can not personally carry out the provisions of this act pertaining to him, the owner, operator, or superintendent shall authorize him to employ enough competent persons to act as his assistants, who shall be subject to his orders. (1891, June 2, P.L. 176, Art. XII, rule 2.)

75. Stations to be established.- A station shall be established at the entrance to each mine or part thereof, as the case may require, and no workman shall pass beyond the station until the mine or part thereof beyond the station has been inspected and reported safe. The fire boss shall remain at the danger station until relieved by some person authorized by himself or the mine foreman, who shall stand guard until such report is received and shall allow no person to pass without permission from the fire boss. (Id., rule 7.)

76. Mine foreman to examine mine, slopes, etc.- The mine foreman or his assistant shall examine every working place in the mine every working day, while the men of such place are or should be at work; and shall direct that each working place be properly secured by props or timber, that all loose coal or rock be pulled down or secured, and that no person shall work in an unsafe place, unless for making it secure. If the mine is idle 48 hours the mine foreman or his assistant shall examine every working place the day before operations are resumed. The foreman, or a competent person designated by him, shall examine at least once every day all slopes, shafts, main roads, traveling ways, signal apparatus, pulleys and timbering, and see that they are in safe and efficient working condition. (1929, Apr. 22, P.L. 630, sec. 5; 1891, June 2, P.L. 176, Art. XII, rule 13.)

77. Roofs and sides to be properly secured.- Any person having charge of a working place in any mine shall keep the roof and sides properly secured by timber or otherwise, and he shall not do any work or permit any work to be done under loose or dangerous material, except for the purpose of making it secure. (1891, Art. XII, rule 14.)

78. Accumulation of water.- When a place is likely to contain a dangerous accumulation of water, the working approaching such place shall not exceed 12 feet in width; and there shall be constantly kept 20 feet or more in advance, at least one bore hole near the center of the working and sufficient blank bore holes on each side. (1891, P.L. 176, Art. XII, rule 15.)

79. Riding on loaded car forbidden.- No person shall ride upon or against any loaded car, cage, or gunboat in any shaft, slope, or plane in or about a mine or colliery. (Id., rule 16.)

80. Number of persons to be hoisted or lowered at one time.- Not more than 10 persons shall be hoisted or lowered at one time in any shaft or slope. Whenever five persons arrive at the bottom of a shaft or slope in which persons are regularly transported, they shall be furnished an empty car or cage and be hoisted, except in mines where there is a traveling way having

an average pitch of 15° or less and which is not more than 1000 feet long. Where two or more loaded cars are regularly hoisted, 20 persons may be hoisted or lowered at one time, provided 30 or more workmen make such a request in writing to the inspector of the district and in his judgment the hoisting appliances are of sufficient strength. In any coal mine where the hoisting appliances are of insufficient strength to hoist or lower the number of persons named, the inspector has the power to reduce the number. (Id., rule 17.)

81. Signals for ascending or descending.- When a person is about to descend or ascend a shaft or slope, the headman or footman shall inform the engineer by signal or otherwise and the engineer shall return a signal before moving or starting the engine. In the absence of a headman or footman the person about to descend or ascend shall give and receive such signals. (Id., rule 21.)

82. Jumping on cars after signal; person entering car in excess of lawful number to get off.- No person except the man giving the signal shall jump on a car after the signal to start. If any person should enter a car in excess of the lawful number, the headman or footman shall request him to get off, which request must be immediately complied with. Violations of this rule must be reported promptly to the mine foreman. (Id., rule 41.)

83. Duty to hoist empty trip.- An empty trip shall be hoisted in any shaft or slope where the engine has been standing idle for one hour or more before men are hoisted or lowered, and no person shall ascend any shaft or slope when working on the night turn until one trip shall first be hoisted therein. (Id., rule 42.)

84. Coal breakers to be heated; dust removed therefrom.- All coal breakers shall be properly heated in order to prevent injury to the health of persons employed therein, and in breakers where coal-dust is so dense as to be injurious to the health of such persons, the owner, operator, or superintendent shall, upon request of the inspector, immediately adopt measures for the removal of the dust as far as practicable. (Id., rules 23, 53.)

85. Cutting of props and timber prohibited.- No person working in a coal mine or colliery shall cut any props or timbers while they are in position to support the roof or sides. When it is necessary for mining coal to remove or to dislodge any of the props or timbers, it must be done by blasting. (Id., rule 55.)

86. Willful damage; acts endangering persons and property.- Any person who shall willfully damage or without authority remove or render useless any fencing, means of signalling, apparatus, instrument or machine, or throw open or obstruct any air way, or open a ventilating door without afterward closing it, or enter a place in a mine against caution, or carry fire, open lights, or matches in places where safety lamps are used, or handle without authority

or disturb any machinery or cars, or do any other act whereby the life or health of any person or the security of property in or about a mine is endangered shall be guilty of an offense against this act. (Id., rule 35.)

87. Abstract of rules to be posted.- For the purpose of making known the rules and provisions of this act to all persons employed in or about a mine to which this act applies, an abstract of the act and rules shall be posted in some conspicuous place at or near the mine or colliery where they may be conveniently read. Any person who pulls down, injures, or defaces such abstract shall be guilty of an offense against this act. (Id., rule 54.)

Article 9. Equipment of Mines and Miners

88. Telephones; means of signaling.- The owner, operator or superintendent of any coal mine or colliery worked by shaft or slope shall maintain a telephone or suitable appliance by which conversation can be held between persons at the bottom and at the top of the shaft or slope, and an efficient means of signaling from the bottom of such shaft or slope to the engineer in charge of the hoisting engine; and shall maintain a telephone by which conversation can be had between persons on the outside of the mine and at the principal centers of operations in the mines. (1929, Apr. 22, P.L. 630, sec. 1.)

89. Handrails; safety catches; gates at landing; safety headgear.- Handrails and efficient safety catches shall be attached to, and a sufficient cover overhead provided on, every cage used for lowering or hoisting persons. The owner, operator, or superintendent of all mines shall see that all landings of shafts are so far as practicable provided with gates, approved by the mine inspector, and operated so that they can be opened only when the shaft carriage is at that particular landing. No man may be employed as a footman at a shaft hoisting coal or other material unless he has provided himself with a safety headgear or helmet of a design approved by the department of mines, which must be worn by him at all times while on duty. (Id., sec. 2.)

90. Protectors to cages and gunboats.- Wherever practicable every cage or gunboat used for lowering or hoisting persons in a slope shall be provided with a protector so constructed that persons thereon shall not be struck by anything falling or rolling down the slope. (1891, June 2, P.L. 176, Art. IV, sec. 11.)

91. Ropes, catches, links, and chains.- The ropes, safety catches, links, and chains shall be carefully examined, every day that they are used, by a competent person delegated for that purpose, and any defects found shall be immediately remedied. The main link of the chain connecting the rope to the cage, gunboat, or car in any shaft or slope shall be made of the best quality of iron; bridle chains made of the same quality of iron shall be attached to the main link, rope or rope socket from the crosshead of the cage or gunboat, when persons are being lowered or hoisted thereon. (Id., secs. 12, 13.)

92. Drums to have efficient brakes; flanges and indicators to be provided.- An efficient brake shall be attached to every drum used for lowering or raising persons or material. Flanges or horns of sufficient dimensions to prevent the rope from slipping off the drum shall be provided, and all machines used for lowering or hoisting persons shall be provided with an indicator to show the position of the cage, car, or gunboat in the shaft or slope. (Id., secs. 14, 15.)

93. Safety lamps.- There shall not be allowed or used in parts of mines in which danger is imminent from explosive gases, lights or fire, other than a locked safety lamp or a locked safety lamp and electric lamp of a type approved by the Department of Mines. Whenever such lamps are required they shall be the property of the owner of the mine. A competent person appointed for the purpose shall examine every safety lamp and electric lamp immediately before it is taken into the workings for use. Such lamps may not be used until they have been examined and found safe, clean, and securely locked, unless permission be given by the mine foreman to have the safety lamps used unlocked. (1929, Apr. 22, P.L. 630, sec. 3.)

94. Keys for safety lamps.- No one except an authorized person shall have in his possession a key or contrivance for unlocking a safety lamp in any mine where locked safety lamps or electric lamps are used. Persons working in a mine or portion thereof where such lamps are used shall not carry into the mine matches, smokers' articles, or anything else which may cause fire. The mine foreman may cause a search of the men for such articles. (Id., sec. 4.)

95. Washhouses to be provided.- It shall be the duty of the owner, operator, or superintendent of each mine or colliery, on written request of 20 or more men employed in any of the mines, to provide a suitable building, not an engine or boiler house, convenient to the principal entrance of such mine, for the use of persons employed therein for the purpose of washing themselves and changing their clothes when entering and returning from the mine. Such building shall be maintained in good order, be properly lighted and heated, supplied with pure cold and warm water, and provided with facilities for persons to wash. Any person failing to comply with this provision, or maliciously injuring said building or any of its appliances, or doing any act tending to the injury or destruction thereof, shall be deemed guilty of an offense against this act. (1891, June 2, P.L. 176, Art. VI, sec. 1.)

Article 10. Shafts, Slopes, Openings, and Outlets

96. Structure to be erected to sustain pulley.- Over all shafts being sunk a safe and substantial structure shall be erected to sustain the sheaves or pulleys, at a height of not less than 20 feet above the tipping place, and the top of such shaft shall be arranged so that no material can fall into the shaft while the bucket is being emptied. The structure shall be erected as soon as a substantial foundation is obtained, and in no case shall a shaft be sunk more than 50 feet without such a structure. (1891, June 2, P.L. 176, Art. IV, secs. 16, 17.)

97. Manner of raising coal and rock.- Rock and coal from shafts being sunk shall not be raised except in a bucket or on a cage, which must be connected to the rope or chain by a safety hook, clevis, or other safe attachment. If provision is made to land the bucket upon a truck, the said truck shall be so constructed that material can not fall into the shaft. (Id., secs. 18, 19.)

98. Shaft guides and attachments; casing or lining.- Such shafts shall be provided with guides and guide attachments applied so as to prevent the bucket from swinging, and such guides and attachments shall be maintained not more than 75 feet from the bottom until its sinking shall have been completed, but this shall not apply to shafts 100 feet or less in depth. When the strata are not safe every shaft shall be securely cased, lined, or otherwise made secure. (Id., secs. 20, 21.)

99. General rules to be observed in shafts.- The following rules shall be observed, as far as practicable, in all shafts to which this act applies:

(a) After each blast the chargeman must see that all loose material is swept down from the timbers before the workmen descend.

(b) After a suspension of work and after firing a blast in a shaft where explosive gases are evolved, the shaft shall be examined and tested with a safety lamp before the workmen descend.

(c) Not more than four persons shall be lowered or hoisted on a bucket at one time and no person shall ride on a loaded bucket.

(d) When persons are employed on platforms in shafts, the platforms must be properly and safely constructed.

(e) While shafts are being sunk all blasts therein must be exploded by electric battery. (Id., sec. 22.)

100. Duty to provide two openings in connection with every seam or stratum of coal and from every lift thereof.- The owner, operator, or superintendent shall not employ or permit any person to work in any mine unless in connection with every seam or stratum of coal and from every lift worked in such mine there are not less than two openings or outlets, separated by a strata not less than 60 feet in breadth underground and 150 feet at the surface, at which openings safe and distinct means of ingress and egress are always available for persons employed in the mine. It shall not be necessary for the two openings to belong to the same mine if the persons employed therein have safe, ready, and available means of ingress and egress by two openings. This section shall not apply to opening a new mine or any new lift, while being worked for making communication between the two outlets, nor to a mine or part thereof in which the second outlet has been rendered unavailable because of the final robbing of pillars previous to abandonment, so long as in both cases not more than 20 persons are employed therein at one time. The cages and other means of egress shall at all times be available for the persons employed where there is no second outlet. (Id., sec. 1.)

101. Proceedings to secure additional outlet upon intervening lands.- If the above paragraph can not be complied with, the owner, operator, or superintendent may bring court proceedings whereby an additional outlet may be made through or upon intervening lands, paying the owner of such lands damages assessed in the proceeding. (Id., sec. 2.)

102. Provisions for ready escape.- Escapements, shafts, or slopes shall be fitted with safe and available appliances by which persons employed in the mine may readily escape in case an accident occurs deranging the hoisting machinery at the main outlets. (Id., sec. 3.)

103. Separate traveling ways.- In slopes where the angle of inclination is 15° or less, a separate traveling way shall be provided and maintained in a safe condition for travel and kept free from steam and dangerous gases. (Id., sec. 4.)

104. Duty with respect to width of passageway and safety holes.- Every passageway used by persons and also used for transportation of coal or material, shall be of sufficient width to permit persons to pass moving cars with safety, or if this is impracticable, holes of ample dimensions, not more than 150 feet apart, shall be made on one side of the passageway. The passageway and safety holes shall be unobstructed and well drained, and the roof and sides made secure. (1891, June 2, P.L. 176, Art. XII, rule 43.)

105. Safety holes at bottom of slopes, etc.- Safety holes shall be made at the bottom of all slopes and planes, and be kept free from obstruction, to enable the footman to escape readily in case of danger. (Id., rule 49.)

106. No inflammable structures over opening.- No inflammable structure, other than a frame to sustain pulleys or sheaves, shall be erected over the entrance of any mine opening, and no "breaker" or other inflammable structure for the preparation or storage of coal shall be erected nearer than 200 feet to any such opening. This shall not apply to breakers that are now erected, or prohibit the erection of a fan drift for ventilation, or of a trestle for transportation of cars from any slope to a breaker or structure, nor shall it apply to any shaft or slope until the work of development and shipment of coal has commenced. (1891, June 2, P.L. 176, Art. IV, sec. 5.)

107. Safety fences at tops of shafts, etc.- The top of each shaft and slope, if dangerous, or any intermediate lift thereof, shall be securely fenced off by railing or by vertical or flat gates. Every abandoned slope, shaft, air hole, and drift shall be properly fenced around or across its entrance. (Id., secs. 6, 7.)

108. Unused underground entrances, etc., to be fenced.- All underground entrances to any place not in actual course of working or extension shall be properly fenced across the whole width of such entrances, so as to prevent persons from inadvertently entering them. (Id., sec. 8.)

Article 11. Boilers, Machinery, Etc.

109. Examination of boilers.- All boilers used for generating steam in and about mines shall be kept in good order. They shall be inspected by a qualified person once in six months and oftener if needed. The result of such examination shall be certified in writing under oath to the inspector for the district within 30 days thereafter. (1891, June 2, P.L. 176, Art. V, sec. 1.)

110. Proximity of boilers to structures.- No boiler used for generating steam shall be placed under nor nearer than 100 feet to any coal breaker or other structure in which persons are employed in preparation of coal, but this shall not apply to boilers or breakers already erected. (Id., sec. 2.)

111. Safety valves; steam gauges.- Each nest of boilers shall be provided with a safety valve of sufficient area for the steam to escape and with weights or springs properly adjusted. Every boiler house shall be provided with a steam gage properly connected with the boilers, and another gage shall be attached to the steam pipe in the engine house and placed in such position that the engineer or fireman can readily examine them. Such steam gages shall be kept in good order, tested, and adjusted as often as once in every six months, and their condition reported to the inspector in the same manner as the report of boiler inspection. (Id., secs. 3, 4.)

112. Guards required to all machinery, stairs, etc.- All machinery used about the mine, and especially in breakers, such as engines, rollers, wheels, screens, shafting, and belting, shall be protected by covering or railing. The sides of stairs, trestles, and dangerous plank walks in and around the collieries shall be provided with hand and guard railing. This section shall not forbid the temporary removal of a fence, guard rail, or covering for repairs or other operations if proper precautions are used and the guard is replaced immediately thereafter. (Id., sec. 5.)

113. Signal apparatus in breaker.- A signal apparatus shall be established at important points in every breaker, so that in case of accident the engineer can be promptly notified to stop the machinery. (Id., sec. 7.)

114. Qualifications of engineer and oilman; time of oiling.- A sober and competent person, not under 18 years of age, shall be engaged to run the breaker engine, and he shall attend to the engine while the machinery is in motion. No person under 15 years of age shall be appointed to oil the machinery, and no person shall oil dangerous parts of it while it is in motion. (Id., secs. 6, 8.)

115. Meddling with machinery prohibited.- No person shall play with, loiter around, or interfere with any machinery in or about any mine or colliery. (Id., sec. 9.)

Article 12. Ventilation

116. Duty to provide pure air; minimum per person.- The owner, operator, or superintendent of every mine shall provide and maintain a constant and adequate supply of pure air for the same. The minimum quantity produced shall not be less than 200 cubic feet per minute for each person employed in any mine, and shall be as much more as the circumstances may require. (1891, June 2, P.L. 176, Art. X, secs. 1, 3.)

117. Air currents.- The ventilating currents shall be conducted and circulated along the face of every working place in sufficient quantities to dilute, render harmless, and sweep away smoke and noxious or dangerous gases, so that all working places and traveling roads shall be safe and fit to work and travel in. (Id., sec. 4.)

118. Certain mines to be divided into districts.- Every mine employing more than 75 persons must be divided into two or more districts. Each district shall be provided with a separate split of pure air, and the ventilation so arranged that not more than 75 persons shall be employed in any one current or split of air. The inlet and return air passages for a particular district must be separated by a pillar of coal or stone if the thickness and dip of the vein will permit, except where it is necessary to cut through the pillar for ventilation, traffic, or drainage. (Id., sec. 6.)

119. Area of air passages; velocity of currents.- Air passages shall be of sufficient area to allow the free passage of not less than 200 cubic feet of air per minute for every person working therein. In mines generating explosive gases the velocity shall not exceed 450 linear feet per minute in any opening through which the air currents pass, if gauze safety lamps are used, except in the main inlet or outlet airways. (Id., sec. 7.)

120. Crosscuts; closing permanently.- When it becomes necessary to close crosscuts connecting the main inlet and outlet air passages of a district, they shall be substantially closed with brick or other suitable building material, laid in mortar or cement when practicable. In no case shall the air stoppings be constructed of plank, except for temporary purposes. (Id., sec. 8.)

121. Doors to close automatically; attendants.- All doors in any way affecting the ventilation shall be so hung and adjusted that they will close automatically. Main doors shall have an attendant who shall open them for transportation and travel and prevent them from standing open longer than necessary, unless a self-acting door is used which is approved by the inspector of the district. (Id., secs. 9, 10; 1899, Apr. 20, P.L. 65, sec. 1.)

122. Arrangement and construction of main doors; extra main doors.- All main doors shall be so placed that when one door is open, another which has the same effect upon the air current shall be closed and thus prevent any temporary stoppage of the current. The framework of such doors shall be substantially secured in stone or brick, laid in mortar or cement, unless otherwise permitted in writing by the inspector. An extra main door shall be so placed and kept standing open as to be out of reach of accident, and so fixed that it can be immediately closed in event of an accident to the doors in use. (1891, June 2, P.L. 176, Art. X, secs. 11-13.)

123. Air bridges.- All permanent air bridges shall be substantially built of such material and of such strength as the circumstances may require. (Id., sec. 14.)

124. Air measurements; monthly reports.- The quantities of air in circulation shall be ascertained with an anemometer or other efficient instrument. Such measurements shall be made by the inside foreman or his assistant once a week at the inlet and outlet airways, at or near the face of each gangway, and at the nearest cross-heading to the face of the inside and outside chamber or breast where men are employed. The headings shall not be driven more than 60 feet from the face of each chamber or breast, and shall be entered in the colliery report book. A report of the air measurements shall be sent the inspector before the 12th day of each month, for the preceding month, together with a statement of the number of persons employed in each district. (Id., secs. 15, 16.)

125. Recording instruments for speed and pressure.- All ventilators shall be provided with recording instruments by which their speed, or the ventilating pressure, shall be registered for each hour. Such data shall be preserved for three months. (Id., sec. 17.)

126. Who to have charge of ventilation.- The mine foreman shall have charge of all matters pertaining to ventilation, and the speed of the ventilators shall be particularly under his direction. Any superintendent causing the mine foreman to disregard the provisions of this act shall be amenable in the same manner as the mine foreman. (1891, June 2, P.L. 176, Art. XII, rule 3.)

127. Ventilation out of order to be reported.- Any miner or other workman discovering anything wrong with the ventilating current, or with the condition of the roof, side, timber, or roadway or with any other part of the mine such as would lead him to suspect danger, shall immediately report the same to the mine foreman or other person in charge of that portion of the mine. (Id., rule 24.)

128. Furnaces prohibited in certain mines.- It shall not be lawful to use a furnace for ventilating any mine wherein explosive gases are generated. (1891, June 2, P.L. 176, Art. X, sec. 2.)

129. Gas or water in abandoned workings.- All worked out or abandoned parts of a mine in operation, so far as practicable, shall be kept free of dangerous bodies of gases or water. If found impracticable to keep the entire mine free from such accumulations, the mine inspector must be immediately notified. (Id., sec. 5.)

130. Examination of mines generating gases.- In mines generating explosive gases the mine foreman or his assistant shall make a careful examination every morning of all working places, traveling roads, and other places which might endanger the safety of the workmen. Such examination shall be made with a safety lamp within three hours at most before time for commencing work, and a workman shall not enter the mine or his working place until the same is reported safe. Every report shall be recorded without delay in a book kept at the colliery for the purpose, and shall be signed by the person making the examination. Such person shall establish proof of the same by marking plainly the date of the examination at the face of each working place and all other places examined. (1891, June 2, P.L. 176, Art. XII, rules 5, 6.)

131. Abandoned portions of mine to be examined.- All accessible parts of an abandoned portion of a mine in which explosive gases have been found shall be carefully examined by the mine foreman or his assistants at least once a week, and all danger found existing therein shall be immediately removed. A report of said examination shall be recorded in a book kept at the colliery for that purpose and signed by the person making the same. (Id., rule 4.)

132. Manner of removing gas.- An accumulation of gas in mines shall not be removed by brushing where practicable to remove it by brattice. (Id., rule 37.)

133. Ignited gas to be extinguished.- When gases are ignited by blast or otherwise the person igniting them shall immediately extinguish the fire if possible, and notify the mine foreman or his assistant. Workmen must see that no gas blowers are left burning upon leaving their working places. (Id., rule 38.)

134. Persons who shall be employed in mines evolving gases.- No person shall be employed who is not competent to understand the regulations of any mine evolving explosive gases, but this rule will not apply to a section of a mine free from such gases. (Id., rule 56.)

135. Withdrawal of workmen from mine affected by noxious gases.- If the person in charge of the mine or any part thereof finds that by reason of noxious gases prevailing in such mine or part thereof, or of any cause whatever, the mine or any part is dangerous, every precaution shall be used to insure the safety of the workmen; and every workman except such as may be required to remove the danger, shall be withdrawn from the mine, or such part as is dangerous, until the endangered area is examined by a competent person and reported safe. (Id., rule 8.)

Article 13. Props and Timbers

136. Necessary props and timbers to be furnished.- The owner, operator, or superintendent or mine foreman of every mine shall furnish miners all props, ties, rails, and timbers necessary for the safe mining of coal and for the protection of the lives of workmen. Such material shall be suitably prepared and delivered to the workmen as near to their working places as they can be conveyed in ordinary mine cars, free of charge. (1891, June 2, P.L. 176, Art. XI, sec. 1.)

137. Workmen to give notice of want of timbers one day in advance.- Every workman in want of props, ties, rails, or timbers shall notify the mine foreman or his assistant at least one day in advance, describing the material required. In case of danger from loose roof or sides, he shall not cut or load coal until the said props and timber have been furnished and the place made secure. (Id., sec. 2.)

138. Failure to furnish timbers an offense.- A failure to comply with the provisions of this article shall be deemed an offense against this act, and shall be taken to be negligence per se on the part of the owner, operator, superintendent, or mine foreman, as the case may be, in an action for the recovery of damages resulting therefrom. (Id., sec. 3.)

Article 14. Explosives and Blasting Operations

139. Keeping explosives in mines.- Explosives shall not be stored in a mine, and a workman shall not have in any one place more than one keg or box containing 25 pounds, unless more is necessary to accomplish one day's work. Every person who has gunpowder or other explosive in a mine shall keep it in a wooden or metallic box securely locked, which shall be kept at least 10 feet from the tracks where room at such distance is available. (1891, June 2, P.L. 176, Art. XII, rules 26, 27.)

140. Manner of handling explosives in general.- Whenever a workman shall open a box containing explosive, or while in any manner handling the explosive, he shall first place his lamp not less than 5 feet away and in such position that the air current can not convey sparks from it. A workman shall not approach nearer than 5 feet to an open box containing powder with a lamp, lighted pipe or other thing containing fire. (Id., rule 28.)

141. Special rules of manufacturers of explosives to be followed.- When high explosives other than gunpowder are employed, the manner of storing, moving, or in any manner using such explosives shall be in accordance with special rules furnished by the manufacturers of the explosives. Such rules shall be indorsed with his official signature and approved by the owner, operator, or superintendent of the mine. (Id., rule 29.)

142. Charging holes for blasting; charge missing fire.- In charging holes for blasting in slate or rock, no iron or steel pointed needle shall be used, and a tight cartridge shall not be rammed into a hole in coal, slate, or rock with an iron or steel tamping bar, unless the end of the bar is tipped with at least 6 inches of copper or other soft metal. A charge of powder or any other explosive in slate or rock which has missed fire shall not be withdrawn or the hole reopened. (Id., rules 30, 31.)

143. Restrictions concerning firing blasts; warning to be given.- No blasts shall be fired in mines where locked safety lamps are used without permission of the mine foreman or assistant, and before such permission is given the person in charge must examine the place and adjoining places and satisfy himself that it is safe to fire such blast. When a workman is about to fire a blast he shall notify all persons who may be in danger therefrom, and shall give sufficient alarm before and after igniting the match, so that persons approaching may be warned of the danger. (Id., rules 11, 33.)

144. Prohibition against shortening time match will burn.- A person about to explode a blast by the use of patent or other squibs or matches shall not shorten the match, saturate it with mineral oil, nor do anything tending to shorten the time the match will burn. (Id., rule 32.)

145. Examination after each blast.- Before commencing work and after firing every blast the miner working a breast or other place shall enter and examine it, and his laborer or assistant shall not go to the face of such breast or place until the miner has examined it and found it to be safe. (Id., rule 34.)

146. Blaster must be qualified.- No person shall be employed to blast coal or rock unless the mine foreman is satisfied that he is qualified to perform the work with ordinary safety. No one except a practical miner shall charge or fire a blast in the absence of an experienced miner, unless he has given satisfactory eveidence of his ability to do so with safety and has obtained permission from the mine foreman or person in charge. (Id., rules 35, 36.)

Article 15. Operation of Cars and Locomotives

147. Speed of locomotives; alarm at front of train pushed.- The speed of locomotives used in mines shall not exceed 6 miles per hour. An efficient alarm shall be attached to the front end of every train of cars pushed by a locomotive. (1891, June 2, P.L. 176, Art. XII, rule 44.)

148. Restrictions on use of locomotives using fire.- Locomotives propelled by steam, if using fire, shall not be used in an intake airway conveying air to any part of a mine where persons are employed, unless there be sufficient air circulating to maintain a healthy atmosphere. (Id., rule 45.)

149. Coupling or uncoupling cars.- Cars shall not be coupled or uncoupled while in motion except by top or bottom men of slopes, planes, or shafts. (Id., rule 46.)

150. Cars on gravity roads; provisions concerning runner.- When cars are run on gravity roads by brakes or sprags the runner shall only ride on the rear of the last car. When said cars are run by sprags a space not less than 2 feet from the car shall be made on one or both sides of the track wherever it may be necessary for the runner to pass along the side of the moving cars, and such space shall be kept free from obstructions. (Id., rule 47.)

151. Cars to be run by suitable persons.- No miner or laborer shall run cars out of any breast or chamber or on any gravity road unless he is a suitable person employed by the mine foreman for that particular work. No person shall be employed for such work under 16 years of age. (Id., rule 48.)

152. Safety blocks; bumpers on cars.- Safety blocks, or some other device for preventing cars from falling into a shaft or running away on a slope or plane shall be placed at or near the head of every shaft, slope or plane, and maintained in good working order. Mine cars shall have bumpers of sufficient length and width to keep the bodies of the cars separated 12 inches when standing on a straight level road and the bumpers touch each other. (Id., rules 49, 51.)

153. Travel on gravity train prohibited; suspension of traffic.- No person shall travel on any gravity train while cars are being hoisted or lowered thereon. Whenever 10 persons arrive at the bottom or top of any plane on which it is necessary for men to travel, traffic thereon shall be suspended long enough to permit them to reach the top or bottom of the plane. (Id., rule 50.)

Article 16. Arbitration

154. Arbitration of demands by inspector concerning things not covered by existing statutes or rules.- Whenever an inspector finds any mine or anything or practice connected therewith, which in any respect is not covered by this act or by any rule, to be dangerous or defective, or in his judgment tends to bodily injury, he shall give notice thereof in writing to the owner, operator, or superintendent of such mine, stating the particular matter requiring remedy, and may demand that the same be remedied. The owner, operator or superintendent shall have the right to refer the demand to a board of arbitration, and the matter shall be arbitrated within 48 hours of the time such demand be made. The party against whom the award is given shall pay all costs. The said board of arbitration shall be composed of three persons, one chosen by the inspector, one by the owner, operator, or superintendent, and a third by the two thus selected. A decision of a majority of such board shall be final and binding in the matter. (1891, June 2, P.L. 176, Art. XVI, sec. 1.)

Article 17. Accidents

155. Loss of life to be reported to inspector.- Whenever loss of life to an employee occurs in or about a mine or colliery, notice thereof shall be given promptly to the inspector of mines, by the mine foreman, outside foreman or other person having immediate charge of the work at the time of the accident; and when death results from personal injury such notice shall be given promptly after knowledge of it comes to the foreman or person in charge. (1891, June 2, P.L. 176, Art. XIII, sec. 1.)

156. Notices of death and serious injuries.- Notices of deaths or serious injuries resulting from accidents in or about mines shall be made to the inspector in writing, and shall specify the name, age, and occupation of the person killed or injured, and the nature and character of the accident and of the injury caused thereby. (1891, June 2, P.L. 176, Art. XIV, sec. 1.)

157. Inspector to provide against further casualties; investigation of fatal accident.- Whenever death occurs or the lives of persons employed in a mine are in danger from any accident, the inspector shall visit the scene as soon as possible and offer such suggestions as in his judgment shall be necessary to protect the lives and secure the safety of the persons employed. In case of death from such accident and he finds it necessary that a coroner's inquest be held, he shall notify the coroner to hold such inquest without delay, and if the inquest is not held within 24 hours of such notice, the inspector shall institute a further and full examination of the accident. For this purpose he is empowered to compel attendance of witnesses and to administer oaths. The inspector shall make a record of such investigations and accidents and preserve the record in his office. (1891, June 2, P.L. 176, Art. XIII, sec. 2.)

158. Inquest to be adjourned to procure presence of inspector.- An inquest held by the coroner upon the body of a person killed by an explosion or other accident shall be adjourned if the inspector of mines be not present, and the coroner shall notify the inspector in writing of such adjourned inquest and the time and place of holding the same, at least three days previous thereto. (Id., sec. 3.)

159. Notice to inspector of inquest; right to examine witnesses.- Due notice of an intended inquest shall be given by the coroner to the inspector. At any such inquest the inspector and any representative of a party in interest may examine witnesses and read the law governing the case to the coroner's jury. (Id., sec. 4; 1915, June 1, P.L. 712, sec. 5.)

160. Notice to inspector of neglect or default shown at inquest.- If at any such inquest the inspector be not present and it is shown by the evidence that the accident was caused by neglect or by any defect in or about the mine, which in the judgment of the jury requires a remedy, the coroner shall send notice in writing to the inspector of such neglect or default. (1891, June 2, P.L. 176, Art. XIII, sec. 5.)

161. Qualifications of jurors on inquest.- No interested person nor one employed in the mine in which the accident occurred shall be qualified to serve on a jury impanneled on the inquest, but the coroner shall impanel a majority of the jury from miners who are qualified to judge of the nature of the accident. (Id., sec. 6.)

Article 18. Care of Injured Miners

162. It shall be unlawful to operate any anthracite mine employing 10 or more men unless the mine is provided with a sufficient quantity of linseed or olive oil, bandages, linens, splints, and woolen and waterproof blankets. These articles shall be stored in rooms erected at convenient places in the mine and on the surface, which rooms shall not be less than 8 by 12 feet, and sufficiently furnished, lighted, clean, and ventilated so that therein medical treatment may be given injured employees in case of emergency. The furnishings shall be adequate to accommodate two or more persons in a reclining and sitting posture. (1901, May 29, P.L. 342, sec. 1; 1913, July 26, P.L. 1361, sec. 1.)

163. Duty of foreman and assistants in case of accidents; care of injured persons.- The mine foreman or his assistants in case of injury to an employee while at work in the mines, shall at once visit the scene of the accident, see that the injured man is carefully wrapped in woolen blankets, removed to the "medical room," and treated with oils or other remedies. He shall also see that the injured man is afterwards carefully wrapped up and sent to the surface, to be taken home in an ambulance or to the mining hospital without expense to the patient. (1901, May 29, P.L. 342, sec. 2.)

164. Application of bandages, etc.; records.- Where accident to any employee involves injury to limbs or loss of blood the foreman or his assistants shall see that the bandages, splints, and linen are applied where necessary to prevent loss of blood and relieve pain, and shall see that the injured person is sent to the surface without delay. The foreman shall keep a book showing required articles on hand, name of persons injured, nature of injury, treatment, and by whom treated at time of accident. (Id., sec. 3.)

165. Inspection of medical rooms.- The mine inspector shall visit each of the medical rooms in his district at least once in six months, see that the law is complied with and examine records of the room. He shall notify the county coroner of any neglect or noncompliance with the provisions of this act, which information shall be regarded as evidence on any inquest held on employees dying from injuries received while working in such anthracite mine. (Id., sec. 4.)

166. Neglect or refusal to comply with act a misdemeanor.- Neglect or refusal to perform the duties required by this act, or violation of its requirements, is a misdemeanor, punishable by a fine not exceeding $500 or imprisonment not exceeding six months or both. (Id., sec. 5.)

167. Right of action for injuries to miners.- For any injury to employees occasioned by violation of this act by any owner, operator, or superintendent of any coal mine or colliery, a right of action shall accrue to the party injured for any direct injuries he may have sustained thereby. In case of loss of life, limb, or bodily power, by reason of such neglect or failure, a right of action shall accrue to the person, widow, or lineal heirs, for the recovery of damages sustained. (Id., sec. 6.)

168. Terms used in act defined.- The term "coal mine" as herein used (paragraphs 162-168), includes the shafts, slopes, drifts, or inclined planes, connected with the excavations penetrating coal stratum or strata, which excavations are ventilated by one general air current or division thereof, and connected by one general system of mine railroads, over which coal may be delivered to one or more parts outside the mine. The term "mine foreman" means the person who shall have, on behalf of the operators, immediate supervision of a coal mine. The term "operator" means any firm, corporation, or individual operating any coal mine. The term "anthracite mine" shall include any coal mine not now included in the bituminous boundaries. (Id., sec. 7.)

169. Motor ambulances and stretchers required.- The owner, operator, or superintendent of every mine or colliery except as hereinafter provided shall keep at such mine a motor ambulance and at least two stretchers, for conveying to their homes any persons injured while at work in the mine. The ambulance shall be constructed on substantial and easy springs, shall be covered and closed, and shall have windows on the sides or ends. It shall be large enough to convey at least two injured persons with two attendants, and shall be provided with spring mattresses or other comfortable bedding, to be placed on roller frames, together with sufficient covering and protection for convenient movement of the injured. It shall also be provided with seats for the attendants. The stretchers shall be constructed of such material and in such manner as to afford the greatest ease and comfort for the carriage of injured persons. The ambulances shall at all times be properly heated. (1917, July 19, P.L. 1124.)

170. Removal of injured persons.- Whenever any person employed in or about a mine shall receive such injury as would render him unable to walk to his home, the owner, operator, or superintendent shall immediately cause such person to be removed in said ambulance to his home or to a hospital, as the case may require. (Id.)

171. Mines excepted.- Any mine or colliery shall be excepted from the requirement of a motor ambulance if the places of abode of all the workmen be within a radius of one-half mile from the principal entrance to such mine, or if less than 20 persons are employed therein. (Id.)

172. <u>When one ambulance may serve several mines</u>.- Where two or more mines or collieries are located within 4 miles of each other, or a motor ambulance is located within 4 miles of each colliery, but one ambulance shall be required, if the said mines or collieries have ready and quick means of communication by telegraph or telephone. (Id.)

173. <u>Use of railways to convey injured; free registration of ambulance</u>.- In case the distance from any mine or colliery to the place of abode of the person injured is such as to permit his conveyance to his home or to a hospital more quickly and conveniently by railway, such mode of conveyance shall be permitted, but in such case the conveyance must be under cover and the comfort of the injured person provided for.

There shall be furnished, free of charge, by the State Highway Department a registration certificate and two number tags for every such ambulance. (Id.)

<u>Article 19. Civil and Criminal Liability</u>

174. <u>Courts may prohibit working of mines contrary to act</u>.- Upon application of the inspector of mines, courts of law or equity shall prohibit the working of any mine in which any person is employed or permitted to be for working in contravention of the provisions of this act. Written notice of intention to apply for such injunction must be given to the owner, operator, or superintendent of the mine or colliery not less than 24 hours before the application is made. (1891, June 2, P.L. 176, Art. XV, sec. 1.)

175. <u>Conviction or acquittal not to be evidence in action for damages</u>.- No conviction or acquittal under this act shall be received in evidence upon the trial of any action for damages arising from the negligence of any owner, operator, or superintendent or employee in any mine or colliery. (Id., Art. XVII, sec. 7.)

176. <u>Proceedings on complaint of citizen; penalty</u>.- Any citizen by affidavit alleging a violation of any of the provisions of this act may cause proceedings to be instituted against the person accused. Convictions are punishable by a fine not exceeding $500, in all cases not otherwise provided for in the act, or imprisonment for not exceeding three months, or both. (Id., sec. 1.)

177. <u>Penalty for violation of duty by inspectors; disposition of fines</u>.- For any violation of duty by the mine inspector prescribed by this act he shall be deemed guilty of a misdemeanor and upon conviction be sentenced to pay a fine of not more than $300 or to be imprisoned for not exceeding three months, or both. All fines imposed under this act shall be paid into the county treasury for the use of the county. (Id., secs. 5, 6.)

Article 20. Pollution of Streams

178. Discharge of coal, culm, or refuse into stream prohibited.- It shall be unlawful for any person, partnership, or corporation to place or discharge or permit to be placed or discharged, into any of the running streams of this State, any anthracite coal, anthracite culm, or refuse from an anthracite coal mine, or to deposit any such coal, culm, or refuse upon the banks of such stream where the same would be likely to slide into or be washed into such stream. This act shall not apply to waters pumped or flowing from coal mines where the coal or culm or refuse have been removed therefor, or prevent the discharge of sewerage from any public sewer system owned or maintained by any municipality. Violations of this act constitute a misdemeanor punishable by a fine of $100 for each offense and a further fine of $50 per day for each day the offense continues, or imprisonment not exceeding one month, or both. (1913, June 27, P.L. 640, sec. 1, 2.)

Article 21. Sale of Coal to Certain Institutions

179. Sale to certain institutions required.- It shall be the duty of the owner, operator, or other person having supervision over any anthracite coal mine or mining operation, upon and after reasonable notice and demand, to furnish and sell at reasonable and current or market prices and at reasonable times such amount of anthracite fuel coal as shall be reasonably necessary for the proper heating of any hospital, poorhouse, or other charitable institution, school building, church, or place of religious worship, or public building, municipal or quasi municipal in character, located within the limits of the municipality in which the coal mining operation is conducted. The owner or operator may demand that the coal so furnished be paid for at the mine or coal pocket where it is furnished or delivered. Violations of this act constitute a misdemeanor punishable by a fine of not more than $1000 or imprisonment for not more than one year, or both. (1923, May 25, P.L. 453.)

Article 22. Prevention of Caving, Collapse of Structures, Etc.

180. Mining so as to cause collapse of certain structures or places prohibited.- It shall be unlawful for any owner, operator, or other person in charge of any anthracite coal mine or mining operation, so to conduct the operation of mining anthracite coal as to cause the caving in, collapse, or subsidence of: (a) Any public building, including churches, theatres, hotels, etc.; (b) any street, bridge, or other public passageway; (c) any track right of way, pipe, or other facility used in the public service by a municipal or public service corporation; (d) any structure used as a human habitation, or any industrial or mercantile establishment in which human labor is employed; or (e) any cemetery or public burial ground.[8] (1921, May 27, P.L. 1198, sec. 1.)

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8 The act of May 27, 1921 (paragraphs 180-184), was held unconstitutional by the United States Supreme Court in Pennsylvania Coal Co. vs. Mahon, et al., 260 U. S. 393.

181. Maps and plans of workings or excavations.- Owners, operators, etc., engaged in mining anthracite coal shall make a true and accurate map of the workings of such coal mine. Such maps shall show in detail all workings which are intended to be developed within the succeeding six months, are required to be deposited with designated public officials, and are open to public inspection. No mining shall be done which is not shown on the map filed at least 10 days previously. All pillars shall be designated by numbers and indicated with corresponding numbers on such maps. (Id., secs. 2, 3.)

182. Access to mines by municipal officers; preventing mining in violation of act.- Designated public officials and their agents shall at all reasonable times be given access to any portion of any anthracite mining operation for determining whether the provisions of this act are being complied with. Such officials may prevent mining in violation of the act. (Id., secs. 4, 5.)

183. Exceptions from operation of act.- The provisions of this act shall not apply to townships of the second class, nor to any area wherein the surface overlying the mining operation is wild or unseated land, nor where such surface is owned by the owner, or operator of the underlying coal and is distant more than 150 feet from any improved property belonging to any other person. (Id., sec. 6.)

184. Violations of act; partial invalidity.- Violations of the act are a misdemeanor punishable by a fine not exceeding $5000 or imprisonment not exceeding one year, or both. The courts of common pleas shall have power to award injunctions to restrain violations. It is declared that the provisions of the act are severable and if the act be declared unconstitutional so far as relating to one or more words, phrases, clauses, or sections thereof, such judicial determination shall not affect any other provision of the act. (Id., secs. 7, 8, 10.)

185. Bureau of mine inspection and surface support.- Boroughs within the limits of the anthracite region may create a bureau of mine inspection and surface support, consisting of two practical mining engineers, appointed by the burgess with the consent of the council, and such assistants, clerks, and employees as the council may provide. (1927, May 4, P.L. 519, Art. XI, sec. 1155, 1156.)

186. Inspection of mines.- Members of the bureau may examine and survey any mine within the limits of the borough at all reasonable times, without obstructing its operation. The owners, operators, or superintendents shall furnish the means necessary for such examination or survey. (Id., sec. 1157.)

187. Operators to furnish maps; contents.- The owner, operator, or superintendent of every coal mine or colliery, within three months after the creation of such a bureau, shall furnish it an accurate map of the workings of such mine, on a scale of 100 feet to the inch, which shall exhibit the workings

in every seam of coal on a separate sheet, and connecting tunnels and passages. It shall show the general inclination of the strata with any material deflection and the tidal elevations of the bottom of every shaft, slope, tunnel, and gangway and of other points deemed necessary by the bureau. The map shall show the number of the last survey station and date of each survey on the gangways or the most advanced workings. Extensions shall be placed upon the map of the bureau at least once every three months. (Id., secs. 1158, 1159.)

188. Certain surface supports not to be removed; penalty.- It shall be unlawful for any person, association, etc., to mine or remove the coal, rock, earth, or other materials forming the natural support of the surface beneath the public highways, streets, alleys, courts, and places of any borough in the anthracite region to such an extent as to remove the necessary support of the surface, without having first constructed an artificial permanet support sufficient to preserve the stability and the surfaces of such public highways, etc. Violations of this act are punishable by imprisonment for not more than 90 days or fine not exceeding $1,000, or both. (Id., secs. 1160, 1161.)

CHAPTER III - BITUMINOUS-COAL MINES

189. Introduction; mines affected by this chapter.- This chapter is taken mainly from the act of June 9, 1911 (P.L. 756), and its amendments, regulating bituminous-coal mining in Pennsylvania. By its terms this act applies only to such mines employing 10 or more persons inside the mine in any one period of 24 hours. Articles 4 and 7, however, are taken from independent enactments and, therefore, so far as the provisions contained in those articles are concerned the limitation to mines employing 10 or more persons does not apply. Attention is invited to the definition of "bituminous mines," appearing in the following paragraph. Should a mine, or portion thereof, that has at any time generated explosive gas in quantities sufficient to be detected by an approved safety lamp, not generate explosive gas during a period of one year, then such mine or portion is not controlled by the provisions of this act for mines or portions of mines generating explosive gas; nor do such provisions apply to any mine wherein explosive gas is being generated only in live entries. (1911, June 9, P.L. 756, Art. XXVIII.)

190. Id.; definitions.- The act of June 9, 1911, provides that for the purposes of the act, the terms and definitions contained therein shall be as follows:

"Mine" includes the shafts, slopes, drifts, or incline planes connected with excavations penetrating coal strata, ventilated by one general air current or divisions thereof, and connected by one general system of mine railroads, when such is operated by one operator.

"Excavations and Workings" includes all the excavated portions of a mine, those abandoned as well as places being worked; all underground workings and shafts, tunnels, and other ways and openings, and all such shafts, etc., in the course of being sunk; and all roads, appliances, and material connected with the same below the surface.

"Shaft" means a vertical opening through the strata, used for ventilation, drainage, or for hoisting men or material.

"Slope" means an incline or opening used for the same purpose as a shaft.

"Operator" means any firm, corporation, or individual operating any coal mine or part thereof.

"Superintendent" means the person having, on behalf of the operator, immediate supervision of one or more mines.

"Mine foreman" means the person whom the operator or superintendent shall place in charge of the inside workings of the mine and of the persons employed therein.

"Inspector" means the person commissioned by the governor to have supervision of mines.

"Bituminous mines" shall include all coal mines in the State not now included in the anthracite boundaries, and whenever the term "mine" appears in this act it shall be construed to mean "bituminous-coal mine."

"Approved safety lamp" shall mean any bonneted safety lamp approved by the Department of Mines. (1911, June 9, P.L. 756, Art. I, sec. 1.)

Article 1. Inspectors and Inspection Districts

191. Division into districts; assignment of inspectors.- The bituminous counties of the State shall be arranged by the secretary of mines into 25 inspection districts, and he shall assign the inspectors to their respective districts and designate their places of abode at points convenient to the mines of their districts. With the governor's consent the secretary may, at any time, redistrict the bituminous districts and add to the number of inspectors. (Id., Art. XXII, secs. 1, 2.)

192. Mine inspectors examining board; personnel.- The mine inspectors examining board for the bituminous-coal mines of Pennsylvania shall consist of the superintendent of public instruction, ex officio, the secretary of mines, who shall be chairman, two mining engineers, and three members who have passed examinations qualifying them as inspectors or mine foremen in bituminous mines generating explosive gas. All members shall be at least 30 years of age, and the last five named must have had 5 years of practical experience in the bituminous mines of Pennsylvania; the latter receive $15 per day while actually engaged on the work of the board. (1929, Apr. 9, P.L. 177, Art. IV, sec. 423.)

193. Qualifications of candidates for office of inspector.- Candidates for office of inspector shall be citizens of Pennsylvania, of temperate habits, of good repute, in good physical condition, and between the ages of 30 and 50 years. They must have a knowledge of the different systems of working coal seams; at least 10 years of practical experience in bituminous mines, the 5 immediately preceding their examination to have been in this State; and practical experience with explosive and other dangerous gases found in coal mines. Upon examination they must give evidence of theoretical and practical knowledge respecting mines and mining and the working and ventilation of mines such as to satisfy the examining board of their capability for the duties imposed upon inspectors. (Id., sec. 3.)

194. Examinations; certificates of qualification.- The principal examination shall be in writing and each applicant shall undergo an oral examination pertaining to explosive gas, safety lamps, methods of ventilation, and mine management. The examination shall not exceed 40 days in duration. Any candidate making a general average of at least 90 per cent shall be deemed successful. The examining board shall certify to the governor and the department of mines the names and percentages of all successful candidates, and a certificate of qualification prepared by the secretary of mines shall be issued to each. (1915, June 1, P.L. 706, sec. 1; Appropriation Acts, 1917, July 16, p. 64, sec. 2.)

195. Commissions of inspectors; salary; bond; persons ineligible.- From the names certified by the examining board the governor shall commission one person to be inspector for each district whose term shall be for four years. The secretary of mines may assign the inspectors to the districts for which in his opinion they are best fitted. Inspectors receive a salary of $4,800

per annum, and must give a bond in the sum of $5,000. No person acting as manager or agent of any coal mine, or as a mining engineer, or who is interested in operating any coal mine, shall at the same time act as an inspector. (1911, June 9, P.L. 756, Art. XIX, secs. 5, 9; 1929, Apr. 9, P.L. 177, Art. IV, sec. 438.)

196. Filling of vacancies; temporary inspectors.- When a vacancy occurs in the office of inspector the governor is authorized to fill the same for the unexpired term, from the list of successful candidates on file in the department of mines. When such list is exhausted the governor shall cause the examining board to hold a special examination, and the board shall certify the names of successful applicants in the same manner as in regular examinations. In case an inspector becomes incapacitated to perform his duties, or is granted a leave of absence, the governor at the request of the secretary of mines shall appoint temporarily to the office a person on the eligible list. (1911, June 9, P.L. 756, Art. XIX, secs. 6, 10.)

197. Duties of inspector; records.- Each inspector shall devote his whole time to his duties. He shall thoroughly examine each mine in his district as often as possible (but at least once every four months), giving special attention to mines generating explosive gas and mines where unusual dangers may be suspected to exist, and see that this act is observed. He shall keep in his office a record of all examinations of mines, showing their condition, especially regarding ventilation and drainage, number of persons employed inside each mine, the extent to which the law is obeyed, and progress made in the improvement of mines. He shall keep a record of all serious accidents, showing the nature and causes thereof and the number of deaths resulting therefrom. (Id., sec. 11.)

198. Right to enter mines; proceedings against violators of act; removal of dangers.- The inspector may at any time enter any mine in his district, or those in other districts when so directed by the secretary of mines, to make examinations or obtain information. He shall institute proceedings against persons violating this act. In case any mine or portion thereof is in the judgment of the inspector in so dangerous a condition as to jeopardize life and health, he shall at once notify the secretary of mines who shall immediately direct two or more inspectors to accompany promptly the said inspector to the said mine. After investigation if the inspectors agree that there is immediate danger they shall direct the superintendent of the mine to remove forthwith the said dangerous condition. If the superintendent fails to do so, the inspectors shall immediately apply to the court of common pleas for a writ of injunction against the operation of said mine, and the court shall at once proceed to hear and determine the case, and if the cause appear to be sufficient shall issue such writ. If any inspector finds such dangerous conditions existing in a mine that in his opinion any delay in removing the workmen might cause loss of life or injury, he shall have the right to withdraw temporarily all persons from such dangerous places until the foregoing provisions can be carried into effect. (Id., sec. 13.)

199. Review of decision by other inspectors; appeal.- The inspector shall exercise sound discretion in the performance of his duties. If the operator, superintendent or other person employed in or about a mine shall be dissatisfied with a decision of the inspector, it shall be his duty to appeal to the Secretary of Mines, who shall direct two or more other inspectors to accompany the inspector of the district to make further examination into the matter in dispute. If the inspectors agree with the decision of the inspector of the district, their decision shall be final, unless within seven days an appeal is taken to the court of quarter sessions. In case of such appeal the court shall forthwith appoint a commission of five persons as required by sections 1 and 2 of article 10 of this act (paragraph 323), and thereafter the proceedings shall be as prescribed therein. (Id., Art. XX, secs. 1, 2.)

200. Inspector's reports; posting; contents; stopping work in unventilated places.- After final examination of any mine, the inspector shall make a report of its condition and post the same in the office of the mine or in some other conspicuous place, where it shall remain for one year open to examination by any person employed in or about the said mine. The report shall show the date of the inspection, the number of cubic feet of air in circulation, where measurement of the air was made, and quantity of air measured at the last cut-through in each split, together with the number of persons employed in each split and at any other place requested by the secretary of mines. The report shall contain such other information as the inspector may deem necessary. If the inspector discovers any room, entry, airway, or other working places being driven in advance of the air current, contrary to this act, he shall order the workmen in such places to cease work at once, until the law is complied with. (Id., Art. XIX, sec. 12.)

201. Monthly and annual reports to the secretary of mines.- Inspectors are required to make monthly report to the secretary of mines of: (a) All fatal and serious accidents in his district, stating the date, nature, and cause of each, and placing the responsibility therefor, together with the name, age, occupation, and nationality of each person killed or injured, and whether married or single and the number of widows or orphans left; which report, or a synopsis of the report, shall be included in the annual report of the Department of Mines; (b) a report giving the name of operator, name and location of each mine inspected, date of inspection, condition of mine, quantity of air in circulation at all points required by the secretary, and the number of persons employed in each split of air. Inspectors shall make annual reports briefly recapitulating the duties performed during the preceding year, and describing the condition of the mines in his district relative to ventilation, drainage, and general sanitary arrangements as relating to the health, safety, and welfare of employees, and containing such suggestions or information of importance as may be deemed necessary or as required by the secretary of mines. (Id., sec. 14.)

202. Annual reports of operators to inspectors.- On or before January 25th in each year the operator or superintendent of every mine shall send to the inspector of the district a correct report specifying, with respect to the preceding year, the name of the operator and officers of the mine, amount of coal mined and coke manufactured, number of different employees, classified, and the total number of days worked during the year. The report shall be in such form and give such information regarding the mine as may from time to time be prescribed by the secretary of mines. Failure to comply with this requirement is a misdemeanor. (Id., Art. XXIII, sec. 1.)

203. Removal of inspectors.- Upon petition of 15 reputable citizens, miners or mine operators, with affidavit of one or more attached, setting forth that any inspector is neglectful, incompetent, or guilty of malfeasance, the court of common pleas shall issue a citation to the said inspector to appear, on not less than 15 days notice, and investigate the charges. If they are sustained the court shall declare the office vacant and so certify to the governor, who shall fill the vacancy; in such event the charges are imposed on the inspector - otherwise, on the petitioners. (Id., Art. XXI, secs. 1, 2.)

Article 3. Maps and Plans

204. Maps of mines to be made; Requisites of.- The operator or superintendent of any bituminous-coal mine shall make, or cause to be made by a competent mining engineer or surveyor, an accurate map of the mine, on a scale of not less than 200 feet to the inch, which map shall show as follows:

1st. All the openings, excavations, shafts, slopes, drifts, tunnels, planes, main entries, cross entries, and rooms and the name or number of each.

2d. An accurate delineation of the boudary lines between the mine and all adjoining mines or coal lands, and the relation and proximity of the workings of the mine to all adjoining mines or coal lands; and, if requested by the inspector, the blue print in the mine office shall show by arrows the direction of the air current in the mine, each split to show in different color in pencil.

3d. The elevation above or below mean tide at Sandy Hook of the top and bottom of each shaft and slope, of all drifts, tunnels, planes, and of the faces of entries adjacent to boundary lines between such mine and any adjoining mine or mines at points not more than 300 feet apart; also the number of last survey station and date of such survey on the entries as they are represented on the map; the location of streams, rivers, lakes, dams, or any other bodies of water on the surface, with their elevations; the location and elevation of any body of water dammed in the mine, or held back in any portion of the mine, giving the true area of the body of water, unless inaccessible before the passage of this act; the location of all boreholes penetrating the coal strata; and the location of all oil and gas wells and oil and gas pipe lines. For the purpose of this paragraph the owner or owners of oil and gas wells and pipe lines shall furnish, at their own expense, to the operator of the mine on which the wells are located or lines constructed, a survey showing the location thereof, within 60 days after the construction or location of wells and pipe lines. (Id., Art. II, sec. 1.)

205. Information as to proximity of workings to adjoining mines.- When the workings of a mine are within 300 feet of the boundary lines between such mine and any adjoining mine, application shall be made by the operator or superintendent to the inspector for informations as to the proximity of the workings of such adjoining mine, and if the workings of the adjoining mine are within 300 feet of such boundary line the inspector shall so notify the operator or superintendent, who shall have such portion of the workings of the adjoining mine surveyed and shown on the map of the mine first mentioned. For such purpose only the engineer or surveyor of any mine shall have the right of entry into any adjoining mine on the written authority of the inspector. (Id., sec. 2.)

206. Copy of map in mine office.- A true copy of the map shall be kept in the office at the mine for the use of mine officials and the inspector, and for inspection of any person working in the mine fearing that any working place is becoming dangerous by proximity to other workings that may contain dangerous accumulations of water or noxious gases. (Id., sec. 3.)

207. Notice of new excavations; approach to water or gas.- At least once every six months the operator or superintendent of every mine shall cause to be shown accurately on the original map and the copy in the mine office, all excavations made during the time elapsed since such excavations were last shown thereon. The operator or superintendent shall order any portion of a mine to be surveyed and entered on the original map, at the request of the inspector when in his opinion such portion is approaching accumulations of water or noxious gases. Whenever any of the workings or excavations shall be driven to their destination, the operator or superintendent shall cause the mining engineer or surveyor to check his previous work so that he can certify that the said map shows correctly all the excavations made therein. (Id., sec. 4.)

208. Inspector's copy of map; exchange of copies by operators.- The inspector shall be furnished a true copy of the aforesaid original map on tracing cloth. Every six months the inspector shall return said copy to the operator or superintendent who shall place or cause to be placed thereon all the extensions made, and all portions of the mine worked out or abandoned during the preceding six months, and shall return the map within 30 days. In lieu of such map on tracing cloth the operator or superintendent may furnish every six months a blue print showing the complete workings of the mine to date. When more than one seam of coal is being worked, the inspector shall be provided with separate maps of each seam. The inspector shall not allow copies of maps in his possession to be made or any information therefrom to be given to any person without consent of the operator. When one mine is working a seam of coal under another operated by a different operator, such operators shall exchange with each other copies of their respective mine maps, showing such portions of their respective mines as may be directly above or below the other mine. (1925, Apr. 7, P.L. 178, sec. 1.)

209. Worked-out or abandoned mine to be shown on map.- Whenever a mine is worked out or abandoned the operator or superintendent shall within 60 days send to the department of mines a tracing of the complete original map, which shall be kept in the department as a public document. The mining engineer or surveyor shall certify to the correctness of the tracing. (1911, June 9, P.L. 756, Art. II, sec. 6.)

210. Survey by inspector.- If an inspector shall have reason to believe any mine map furnished him is inaccurate, he may have made a survey and a new map. The operator is liable for the cost of the survey and map unless the one furnished by him is shown to be sufficiently accurate. (Id., sec. 7.)

211. Map of workings near division line to be furnished adjoining owner.- Upon written request the owner or operator of each bituminous coal or clay mine shall provide or lend to the owner of adjoining coal or clay lands a true map of such mine, showing all rooms, headings, and workings convenient to and along the division line of such adjoining lands. On failure to comply with such request, or for verifying the accuracy of a map furnished, the owner of such adjoining lands may enter such mines, with or by agents, and ascertain and verify by surveys the true condition, plan, and workings of such mines, convenient to and along the division line. Five days notice shall be given to the operator of such mines before entering the same, and the time selected shall be such as shall least interfere with the active operations of the mines. (1911, June 15, P.L. 954, sec. 1.)

Article 3. Duties of Mine Superintendent in General

212. Supplies required; examination of mine record book.- Every superintendent shall keep on hand at each mine a sufficient quantity of all materials and supplies required to preserve the health and safety of employees. If necessary materials or supplies can not be procured, he shall at once notify the mine foreman, who shall withdraw the men from the mine or portion of mine until the supplies are received. The superintendent shall, at least once a week, examine carefully and countersign all reports entered in the mine record book by the mine foreman. If he finds on such examination that the law is being violated, he shall order the mine foreman to stop said violation forthwith and shall see that his order is complied with. (1911, June 9, P.L. 756, Art. III, sec. 1.)

213. Superintendent to direct compliance with law; when foreman is to act as superintendent.- The superintendent shall not obstruct the mine foreman or other officials in the fulfillment of their duties but shall direct that the foreman and other employees comply with the law. At any mine where a superintendent is not employed, the duties herein prescribed for the superintendent shall devolve upon the mine foreman. (Id., sec. 2.)

214. Danger signals.- The superintendent shall provide a sufficient number of danger signals upon request of the mine foreman, which the foreman or his assistant shall distribute in the mine at convenient places for the use of the fire bosses. Danger signals in all mines shall be uniform and of a design approved by the secretary of mines, and shall be kept in good condition. Defective signals shall not remain in any mine. (Id., sec. 3.)

215. Rules, notices, and record books.- The superintendent shall keep on hand a supply of the printed rules, notices, and record books required by this act, which shall be furnished through the inspector on request. The superintendent shall see that the rules, etc., are delivered to the proper persons, that they are properly cared for, and that the rules and notices are posted in conspicuous places near the mine entrance and kept in legible condition. (Id., sec. 4.)

216. Work near dangerous accumulation of water; barrier pillars between adjoining mines.- The superintendent shall not permit mining of coal within 50 feet of any abandoned mine or portion thereof containing a dangerous accumulation of water, until such water is drained off. He shall not permit the mining of coal in any seam the entire distance to a property boundary line (not including boundaries around reservations or along crop lines), when on the adjoining property there are mine workings in the seam within 3000 feet of the boundary line, but shall leave a barrier pillar of not less than 10 feet plus 2 feet for every foot or fraction thereof of thickness of the bed, plus 5 feet for each 100 feet or fraction thereof of cover over the bed at the boundary line. Where prior to the passage of this act the coal on one side of the line shall have been mined closer to the line than hereinbefore permitted, then the barrier pillar to be left in the mine approaching the boundary line shall be at least equal (when added to that already left in the adjoining mine) to that hereinbefore required on both sides of said property boundary line. If in the opinion of the district mine inspector or the superintendent of either mining property, the barrier pillar is deemed insufficient, then after due notice to the operator of the adjoining mining property, a barrier pillar of unmined coal (one-half on each side of the boundary line, except as provided above) shall be left, of such thickness as in the judgment of the inspector and the superintendent or owner of either mining property, is deemed necessary to afford safety and protection. If it shall be agreed by the superintendents of such adjoining properties that such boundary line is so located that there is no danger to property or lives in mining coal on either or both sides up to said line, mining to the line shall be lawful if all danger from accumulated water and gas shall have first been removed. If any of the parties in interest fail to agree on the carrying out of the provisions of this act, any one of the parties may appeal to the secretary of mines, who shall appoint three reputable and competent mining engineers to constitute a commission to investigate the conditions and determine the necessity for leaving barrier pillars, or a pillar of greater thickness than is provided for in this act, or the adequacy of an existing pillar to withstand a water head, as the case may be. The decision of the commission shall be binding on all parties concerned. (1929, Apr. 10, P.L. 472, sec. 1.)

217. Safety devices and signals on rear of cars.- The superintendent shall provide a safety catch or other safety device to be placed on the rear end of the rear car of full trips being hoisted up slopes, and shall provide suitable signals to be placed on the rear end of the rear car of all trips hauled in the mines by locomotives of any kind. (1911, June 9, P.L. 756, Art. III, sec. 6.)

218. Withdrawal of certificate of foreman or fire boss.- If the mine foreman, assistant mine foreman, or fire boss neglects his duties or is incapacitated to perform his duties and information of same reaches the superintendent he shall make a thorough investigation, and if he finds evidence to sustain the charge shall inform the inspector who shall inform the court of common pleas by petition. Said court shall issue a citation to the person charged and investigate the allegations. If the charges are sustained the court shall notify the Department of Mines and instruct it to withdraw the certificate of the delinquent. He may be reinstated upon passing a satisfactory examination and satisfying the examining board that he has reformed. (Id., sec. 7.)

219. Notice to inspector of certain occurrences.- The operator or the superintendent shall within 30 days send to the inspector notices when a mine has been abandoned or its working discontinued; when work has commenced for opening a new mine; when working of a mine is resumed after abandonment or a discontinuance exceeding two months; and when any change occurs in the name of a mine or the operator. (Id., sec. 8.)

Article 4. Mine Foremen, Assistant Mine Foremen, and Fire Bosses; Examination; Qualifications

220. Boards of examiners; meetings.- On petition of the district mine inspector the court of common pleas in any county in the district shall appoint an examining board of three persons, consisting of a mine inspector, a miner, and an operator or superintendent. The miner shall be in actual practice and have had at least 10 years of practical experience in the bituminous mines of Pennsylvania. Members receive $6 per day while actually employed and are reimbursed for necessary expenses. The secretary of mines shall determine the districts in which the boards of examiners shall meet for holding examinations, and at least two weeks notice of the time and place shall be given. (1923, May 31, P.L. 481, secs. 1, 2, 4.)

221. Committee to prepare questions and answers.- The secretary of mines shall select from the members of the examining boards a committee comprised of two mine inspectors, two operators or superintendents, and two miners, who shall prepare the questions and answers to be used by all the examining boards in the bituminous region. The committee shall distribute the questions and answers in sealed packages to the chairman of the boards, and at the commencement of each session the chairman of each board shall open the package in the presence of all the members. The secretary of mines may at any time convene the committee for preparing questions and answers for special examinations. (Id., sec. 3.)

222. Designation of boards before which applicants are to appear; notice by applicants.- The secretary of mines shall designate the boards before whom applicants in the various districts shall appear. Persons desiring to enter the examination, if possible, shall notify the chairman of their intention 10 days prior to the date of examination. (Id., sec. 5.)

223. Qualifications of applicants.- Applicants for certificates of qualification as mine foremen, assistant mine foremen, and fire bosses shall be citizens of the United States, of good moral character and temperate habits, at least 23 years of age, and shall have had at least five years of practical experience (two years in Pennsylvania), after 16 years of age, as miners or mining engineers, or men of general work inside the bituminous mines of the United States. Graduates in the coal mining course of a recognized institution of learning may be granted certificates if possessed of an aggregate of not less than three years of such practical experience in Pennsylvania. Applicants for certificates as fire bosses shall also have had experience in Pennsylvania bituminous mines that generate explosive gas. All applicants shall be able to read and write the English language, and shall furnish the board with certificates as to their character and temperate habits and length of service in the different mines. (1925, Apr. 7, P.L. 174, sec. 1.)

224. Certificates of qualification.- Certificates of qualification as mine foremen shall be of two grades. First-grade certificates shall be granted to persons giving satisfactory evidence of their ability to perform the duties of mine foremen in gaseous mines and receiving an average of 80 per cent in the examination. Second-grade certificates shall be granted to persons giving such evidence as to nongaseous mines and receiving an average of 80 per cent. Certificates of qualification as assistant mine foremen shall be granted to persons giving satisfactory evidence of their ability to perform duties of assistant foremen in gaseous mines and receiving 70 per cent in the examination. Certificates of qualification as fire bosses shall be granted to persons who have given the board satisfactory evidence of their ability to perform the duties of fire bosses in gaseous mines and who have received an average of 65 per cent in the examination. (Id., sec. 1.)

225. Id.; issue and contents.- Each examining board shall certify to the secretary of mines every person whose examination shall disclose his fitness for the duties of mine foreman, assistant mine foreman, or fire boss; and the secretary shall prepare certificates of qualification and send them to the chairman of the board for distribution. Each certificate shall contain the full name, age, and place of birth of applicant, and the length and nature of his previous service in or about the mines. (1923, May 31, P.L. 481, sec. 8.)

226. Employment of foremen, assistant foremen, and fire bosses.- It shall be unlawful for any operator or superintendent to employ as mine foreman in a bituminous mine, or as assistant mine foreman in a bituminous gaseous mine, any person who has not obtained the proper certificate of qualification, unless such person is, in the judgment of the operator, equally competent with the persons who are holders of such certificates. It shall be unlawful to employ as fire boss in a bituminous mine any person who has not obtained the proper

certificate of qualification unless such person is, in the judgment of the operator, equally competent with holders of such certificates. (Id., sec. 10.)

227. Forging, counterfeiting, or false statements.- Forging or counterfeiting a certificate or making false statements in any certificate is a misdemeanor, punishable by a fine of not less than $200 nor more than $500, or imprisonment for not exceeding one year, or both. (Id., sec. 11.)

228. Assistant mine foremen in certain mines.- Nothing in this article shall prevent a first-grade mine foreman from acting as assistant foreman, or a second-grade mine foreman from acting as assistant foreman in a nongaseous mine. (Id., sec. 12.)

Article 5. Mine Foreman and His Duties in General

229. Duty to employ foreman; assistants; control of employees.- The operator or superintendent shall employ a competent mine foreman for every mine where 10 or more persons are employed, who shall be under the supervision and control of the operator. (As to persons eligible, see paragraph 226.) The mine foreman shall have full charge of all the inside workings and the persons employed therein, subject to the supervision of the operator. When the mine workings become so extensive that the mine foreman is unable personally to carry out the requirements of this act pertaining to his duties, he shall have the right to employ a sufficient number of competent persons to act as his assistants, who shall be under his and the operator's instructions. If the mine is generating explosive gas, in quantities sufficient to be detected by an approved safety lamp, the mine foreman's assistants must possess first-grade assistant mine foremen's certificates, or be persons who, in the judgment of the operator, are equally competent. In case of temporary absence of the mine foreman he may deputize his work to his assistant. The right to hire and discharge employees, the management of the mine and the direction of the working forces, are vested exclusively in the operator, and no person shall interfere or attempt to interfere with such right. (1915, June 1, P.L. 716, sec. 2.)

230. Shall devote whole time to duties; ventilation.- The mine foreman shall devote his whole time to his duties in the mine when it is in operation. He shall keep a careful watch over the ventilation apparatus, the airways, traveling ways, timbering, and drainage, and see that all stoppings along airways are properly built. He shall see that proper cut-throughs are made and that they are closed when necessary so that the ventilating current can be conducted in sufficient quantity through the last cut-through to the face of each room and entry by means of check doors. He shall not permit any room or entry to be turned in advance of the ventilating current or of the last cut-through in the entry, excepting room necks, which may, with the consent of the inspector, be turned by entrymen driving entries. (1911, June 9, P.L. 756, Art. IV, sec. 2.)

231. Props and timbers.- The mine foreman shall see that every working place is properly secured by props or timbers, and that no person is permitted to work in an unsafe place, except for making it safe. He shall see that the workmen are provided with sufficient props, cap pieces, and timbers of suitable size, which shall be delivered at the working faces or as near thereto as they can be conveyed in mine cars. He shall also see that props are cut square at both ends and as near as practicable to the proper length required for the places where they are to be used. Every workman in need of props, cap pieces, and timbers shall notify the mine foreman (or a person delegated by him), one day in advance, describing the material required. In case of emergency the timber may be ordered immediately. If the necessary timbers can not be supplied when required, the mine foreman or his assistant shall instruct the workmen to vacate the place until the timber needed is supplied. (Id., secs. 6, 7.)

232. Shelter holes; entries; travel on slopes, etc.- The mine foreman shall see that on all hauling roads holes for shelter shall be cut into the strata, not less than 2½ feet deep and 4 feet wide and level with the road, at least every 30 yards, and kept whitewashed and clear of obstruction, except in entries from which rooms are driven at regular intervals not exceeding 90 feet, the entrance to each room being kept unobstructed for 3 feet. On all main hauling roads on which hauling is done by machinery, such shelter holes shall be not more than 15 yards apart, except in entries from which rooms are opened at regular intervals not exceeding 45 feet. All shelter holes shall be made on the same side of the entry. All entries driven after the passage of this act shall have a clear space of 2½ feet from the side of the car to the rib, which shall be made and continued throughout on one side, if in the judgment of the inspector the condition of the roof will permit, and shall be kept clear of obstruction. No persons except officials or repairmen shall be permitted to travel on slopes or gravity or incline planes while the cars thereon are in motion. (Id., sec. 8.)

233. Coal to be properly mined; removal of overhead dangers.- The mine foreman shall direct that the coal be properly mined before it is blasted. "Properly mined" means that the coal shall be undercut, center cut, top cut, or sheared by pick or machine, and in any case the undercutting shall be as deep as the holes are laid. In mines generating explosive gas in quantities sufficient to be detected by an approved safety lamp, when the coal seam is 5½ feet or more thick, "properly mined" shall mean that in all entries less than 10 feet wide, wherein the coal is undercut, it shall also be sheared on one side as deep as the undercutting before any holes are charged and fired, or the coal shall be blasted in sections by placing the first hole near the center of the seam. The mine foreman shall direct that the miner set sprags as often as necessary, at a distance not exceeding 7 feet apart, under the breast of undermined or center mined coal. He shall also direct at what hours blasting shall be done in the mine, and a notice of the blasting shall be posted and a copy kept on file at the mine office. The mine foreman shall see that as the miners advance, all dangerous and doubtful pieces of coal, slate, and rock overhead are taken down or carefully secured. Any workman

neglecting or disobeying the instructions of the foreman or his assistant in regard to securing his working place shall be suspended or discharged, and if such negligence or disobedience results in serious injury or loss of life, the mine foreman shall give the name of said workman to the inspector for prosecution. (1929, Apr. 18, P.L. 613.)

234. Dampening dust; rock-dusting.- In portions of a dry and dusty mine where explosive gas is being generated in quantities sufficient to be detected by an approved safety lamp, the mine foreman shall see that the rooms and entries are moistened by water or other efficient means as often as necessary, and that the dust is taken out of the mine as often as necessary. Rock-dust may be substituted for or used in conjunction with water under the following conditions: The rock-dust shall be pulverized so that 100 per cent will pass through a sieve having 20 meshes per linear inch, and 50 per cent through a sieve having 200 meshes. It shall not contain more than 5 per cent combustible matter nor more than 25 per cent of quartz or free silica particles, and must not be unduly absorbent of moisture. Preference shall be given to lighter-colored rock-dust. The rock-dusting shall be done with such frequency that all surfaces required to be rock-dusted shall be kept in such condition that the incombustible content of the adhering dust shall not be less than 55 per cent.

The rock-dust shall be distributed on top, bottom, and sides of all main haulages and all active entries to the last open cut-through, and to a distance of not less than 20 feet in all active rooms and pillars, workings, and in all return airways where hauling or traveling is done. In other places where no hauling is done the rock-dust may be distributed by the air current or other means, provided the amount specified in this act is distributed; or such places may be protected by rock-dust barriers, which shall be subject to the approval of the inspector as to design and location. When such barriers are installed the amount of rock-dust used shall be at least 100 pounds per square foot of average cross section of entry at the barrier zone. The barriers shall be maintained in workable condition and inspected monthly, and a report of such inspection showing the condition of the barrier shall be entered in the book furnished by the department of mines for recording the rock-dust samples.

The superintendent shall see that a sufficient number of samples of dust are gathered each month from the road, roof, rib, and timbers, on each split of air and tested to determine if any part of the mine requires re-dusting, and a record shall be kept in a book provided for that purpose, showing the location at which samples have been taken and the results of the analyses or tests. When the working faces of entries, airways, rooms, and other working places are kept damp by watering, the outbye portions of such places, when required, shall be treated with rock-dust or water.

If in the judgment of the mine inspector any mine or portion thereof is of such a dry and dusty nature as to cause a hazard from coal-dust, he shall direct that the said mine or portion be rock-dusted or watered. Should the operator be dissatisfied with the decision of the inspector in regard to watering or rock-dusting, or to rock-dust barriers, it shall be his duty to appleal to the secretary of mines, who shall direct two or more other inspectors to accompany the inspector of the district to make further examination into the matter. If the said inspectors agree with the decision of the

inspector of the district their decision shall be final unless the dissatisfied person shall appeal to the court of quarter sessions, whereupon the court shall appoint a commission of five persons as required by article 10 of this act (paragraph 323). (1929, Apr. 18, P.L. 613, sec. 1.)

235. Removal of dangers; visits to workings; instructions for mining; reports of assistants.- The mine foreman shall give prompt attention to the removal of all dangers reported to him. In case it is impracticable to remove the danger at once, he shall notify every person whose safety is menaced thereby to remain away from the portion where the dangerous conditions exist. He or his assistant shall once each week travel and examine all the air courses, roads, and openings that give access to old workings or falls, and record the condition of all dangerous places in the book provided for that purpose. The mine foreman shall employ sufficient assistants to insure a visit to each working place, either by himself or by his assistants, once each day while the employees are at work, and shall give special care and attention to the men drawing pillars. He or his assistant shall direct that the holes for blasting be properly placed and shall designate the angle and depth of holes, which shall not be deeper than the undercutting, center cutting, top cutting, or shearing, and the maximum quantity of explosives required for each hole, and the method of charging and tamping. Instructions shall be given the men as to when, where, and how timber shall be placed and also in a general way how to mine coal with safety to themselves and others. At the end of each shift each assistant foreman shall make a report in a book provided for that purpose, giving the general condition as to safety of the working places visited by him during the day. The mine foreman shall read carefully the daily report of each assistant and shall sign the reports with ink not later than the day following. (1911, June 9, P.L. 756, Art. IV, sec. 10.)

236. Removal of gas; fencing dangerous places; danger signals.- The mine foreman shall see that every mine generating explosive gas is kept free of standing gas in all working places and roadways. Any accumulation of explosive or noxious gases in the worked-out or abandoned portions of a mine shall be removed as soon as possible after discovery, if practicable. Persons who may be endangered shall not be allowed in that portion of the mine until the gases shall have been removed. The foreman shall see that all dangerous places and entrances to worked-out and abandoned places are properly fenced off so that no person can enter, and that danger signals are posted upon the fencing to warn persons of the existing danger. Where it has been found impracticable to remove explosive gas from the inaccessible top of a fall, the mine foreman shall report the condition to the superintendent, who shall immediately report to the inspector, requesting him to make a prompt personal investigation. If the superintendent and the inspector are unable to devise means to remove the gas within a reasonable time, the inspector shall direct that a borehole or holes, not less than 6 inches in diameter be drilled from the surface to a high point on the fall, in order to give the gas an opening for escape. (Id., secs. 11, 12.)

237. Employment of inexperienced miners in gaseous mines.- In every mine generating explosive gas, in quantities sufficient to be detected by an approved safety lamp, where coal-dust is being carried in the air currents in quantities indicating danger, the mine foreman shall see that no person is employed to work in the mine until he has given satisfactory proof that he can do the work allotted to him without endangering the lives of his coemployees, unless the person is put to work with an experienced miner, whose duty it shall be to instruct him how to perform his work safely. (Id., sec. 13.)

238. Shot firers in gaseous mine; regulations for blasting.- In such portions of a mine where explosive gas is being generated in quantities sufficient to be detected by an approved safety lamp and in which locked safety lamps are used, the mine foreman shall employ sufficient competent persons, able to speak the English language, to act as shot firers, whose duty shall be to charge, tamp, and fire all holes properly placed. No holes shall be fired by any person other than a shot firer. They shall use none but incombustible material for tamping, which the mine foreman shall see is provided for them at convenient places, and under no condition shall they use coal-dust or other combustible material. All such holes shall be fired by an electric apparatus, and no one but the shot firer shall connect the wires of or operate the apparatus. Each shot firer shall keep a record of and report to the foreman every hole that he has refused to charge, every blown-out shot, and every hole that has misfired. The shot firers and miners who are permitted by this act to fire their own shots shall visit and examine the places where shots have been fired before leaving the mine to see that there is no fire or other danger existing. In all mines in which coal is blasted from the solid, all holes shall be fired when all workmen are out of the mine except the shot firers and other persons delegated by the mine foreman to safeguard property. No shot firer or other person shall fire a shot in any working place if his safety lamp can detect explosive gas at the roof. In gaseous, dusty mines in which approved locked safety lamps are used, he shall fire no holes unless the entries and rooms which are dry and dusty are so thoroughly wetted as to prevent the existence of any dry dust for not less than 80 feet from the hole, unless the dust is rendered inert to explosibility by rock-dust. In mines wherein the coal is being blasted from the solid, the mine foreman shall see that the provisions of this section are complied with. (1929, Apr. 18, P.L. 613, sec. 2.)

239. Temporary suspension of operations; signals; fences.- When operations are temporarily suspended the superintendent and mine foreman shall see that danger signals are placed across the mine entrances. If the circulation of air through the mine be stopped, each entrance to same shall be fenced off in such manner as will ordinarily prevent persons from entering, and a danger signal displayed on said fence. The mine foreman shall see that all danger signals are in good condition, and if any become defective he shall notify the superintendent. (1911, June 9, P.L. 756, Art. IV, sec. 15.)

240. Drainage.- The mine foreman shall see that the water is drained out of the working places before men enter, and that the working places are kept as free from water as practicable during working hours. (Id., sec. 16.)

241. Work in proximity to abandoned mine.- In any working place that is being driven within supposedly dangerous proximity to an abandoned mine that may contain a dangerous accumulation of water or gas, the mine foreman shall see that at least two boreholes shall be maintained not less than 20 feet in advance of the face, and on each side of such working place boreholes of the same depth shall be drilled diagonally not more than 8 feet apart. Any place driven to tap water or gas shall not be more than 10 feet wide. No water or gas from an abandoned mine, or portion thereof, and no boreholes from the surface, shall be tapped until the employees, except those engaged at such work, are out of the mine, and such work shall be done under the immediate instruction of the mine foreman, with the use of locked safety lamps. (1929, Apr. 10, P.L. 472, sec. 2.)

242. Daily and weekly reports; reports of fire boss.- The mine foreman shall daily enter plainly and sign with ink in a book provided for that purpose, a report of the condition of the mine, which shall state any danger that may have come under his observation or have been reported to him by his assistants or the fire bosses. The report shall also state whether or not there is a proper supply of material on hand and whether the requirements of the law are complied with. He shall once each week enter in the book a true report of all air measurements required by this act, designating the place, the area of each cut-through and entry separately, the velocity of the air in each cut-through and entry, and the number of men employed in each split of air, with the date when measurements were taken. The report book shall be kept in the mine office for examination by the inspector, or by any person working in the mine in the presence of the foreman. The mine foreman shall also, each day, read carefully and countersign all reports entered in the record book of the fire bosses. (1911, June 9, P.L. 756, Art. IV, sec. 18.)

243. Reports of accidents.- The mine foreman shall once each week report to the inspector all fatal and serious accidents occurring in or about the mines, giving the age, nationality, and occupation of the injured persons, together with facts as to the families or dependents affected. (Id., sec. 19.)

244. Employment of fire bosses.- The mine foreman shall employ a sufficient number of fire bosses to examine the mines in accordance with sections 1 to 3, article 5, of this act (paragraphs 249-251). The mine foreman or his assistants shall see, as often as practicable, that the fire boss has left his mark in places examined or reported as examined. (Id., sec. 20.)

245. Safety blocks and switches.- The mine foreman shall see that safety blocks or some other device are constructed to prevent cars from falling into the shaft or slope or running away on slopes and incline planes. Safety switches, drop logs, or other devices, shall be used on all slopes and incline planes, and the foreman shall see that such devices are maintained in good working order. (Id., sec. 21.)

246. Locked safety lamps; transportation of tools.- The mine foreman shall see that locked safety lamps are used when and where required by this act. The transportation of tools in and out of the mine shall be under the direction of the mine foreman or his assistant. (Id., sec. 22.)

247. Report of violations of act.- It shall be the duty of the mine foreman to report immediately all violations of this act to the inspector. (Id., sec. 23.)

248. Duties of assistant foremen.- When assistant mine foremen are employed, their duty shall be to assist the mine foreman in complying with this act. In the absence of the mine foreman, they shall perform his duties and shall be liable to the same penalties as the mine foreman for any violation of this act. (Id., sec. 24.)

Article 6. Fire Boss and His Duties in General

249. Duty to employ; competency; duties.- In portions of a mine wherein explosive gas has been generated within one year before the passage of this act, or shall be generated after its passage, in sufficient quantities to be detected by an approved safety lamp, the mine foreman shall employ one or more fire bosses, whose competency to act as such shall be evidenced by a certificate of qualification from the department of mines, or a person who in the judgment of the operator is equally competent with such holders. The fire boss shall first see that the air current is traveling in its proper course and shall then examine carefully, before each shift enters the mine, every working place, all places adjacent to live workings, every roadway and every unfenced road to abandoned workings and falls in the mines. In making the examination he shall use no light other than that enclosed in an approved safety lamp. The examination shall begin within three hours prior to the appointed time to enter the mine. The fire boss shall examine for dangers in all portions of the mine under his charge, and after each examination shall leave at the face and side of every place examined the date as evidence that he has performed his duty. He shall also examine the entrances to all worked-out and abandoned portions adjacent to the roadways and working places where explosive gas is likely to accumulate, and shall place a danger signal across the entrance to every place where explosive gas is discovered or immediate danger is found to exist from any other cause. The meaning of all danger signals shall be explained to the non-English speaking employees of the mines by the mine foreman, assistant foreman, or fire boss, through an interpreter. (1915, June 1, P.L. 716; sec. 3.)

250. Record of examinations; report of safe condition.- A suitable record book shall be kept at the mine office, on the surface of every mine wherein fire bosses are employed. Immediately after making an examination the fire boss shall enter in the book, with ink, a record of such examination and shall sign the entry. This record shall state the time taken in making the examination, and the nature and location of any danger discovered, and if any danger has been discovered the fire bosses shall immediately report the location thereof to the mine foreman. No person shall enter the mine until the fire bosses

return to the mine office on the surface, or to a station located in the intake entry of the mine (where a record book as provided for in this section shall be kept and signed by the person making the examination), and report to the mine foreman or his assistant by telephone or otherwise that the mine is safe for the men to enter. When a station is located in any mine it shall be the duty of the fire bosses to sign also the report entered in the record book in the mine office on the surface. The record books of the fire bosses shall during workings hours be accessible to the inspector and the employees of the mine. (1911, June 9, P.L. 756, Art. V, sec. 2.)

251. Examination during working hours.- A second examination by the same or other fire bosses shall be made during working hours of every working place where men are employed. (Id., sec. 3.)

252. Permanent station; equipment.- The mine foreman and the fire boss shall, at or near the main entrance to the mine, provide a permanent station with a proper danger signal. In every mine generating explosive gas in sufficient quantities to be detected by an approved safety lamp, when the working portions are 1 mile or more from the entrance to the mine or from the bottom of the shaft or slope, a permanent station of suitable dimensions may be erected by the mine foreman (provided the location is approved by the inspector) for the use of the fire bosses, and in the station a fireproof vault of ample strength shall be erected of brick, stone, or concrete in which the temporary record book of the fire bosses shall be kept. No person except the mine foreman, and in case of necessity other persons designated by him, shall pass beyond said permanent station until the mine has been examined by a fire boss and reported by him to be safe. The fire boss shall not allow any other person to enter or remain in any portion of the mine through which a dangerous accumulation of gas is being passed in the ventilating current. He shall report at once any violations of this article to the mine foreman. (Id., sec. 4.)

253. Penalty for passing or removing danger signals; foreman to report violations to inspector.- Any person, except those hereinbefore provided for, who passes any danger signal into the mine or at the entrance to any place in the mine, or who removes such danger signals before the mine has been examined and reported to be safe or without permission from the mine foreman, assistant mine foreman, or the fire boss, shall be guilty of a misdemeanor. The mine foreman having knowledge of said violation shall notify the inspector at once in writing, who shall forthwith enter proceedings against such persons. Any mine foreman failing to notify the inspector of any violation of the provisions of this article, reported to him or coming under his observation, shall be guilty of a misdemeanor. (Id., sec. 5.)

254. Failure of fire boss to comply with act.- Any fire boss who neglects to comply with the provisions of this article or who shall make a false report of the condition of any place allotted to him for examination, shall be guilty of a misdemeanor, and shall be suspended by the mine foreman and his name given to the inspector for prosecution. If found guilty he shall return his certificate of qualification as fire boss to the department of mines. After the expiration of six months, he may again be an applicant for a certificate at any regular examination, but if found guilty of a second offense shall not be eligible for reexamination. (Id., sec. 6.)

255. Foreman or assistant acting as fire boss; fire boss acting as foreman in emergency.- Nothing in this article shall prevent a first-grade mine foreman or a first-grade assistant mine foreman from acting as fire boss, or a regularly employed fire boss from acting in an emergency as a first-grade assistant mine foreman. (Id., sec. 7.)

Article 7. Cars; Weighing and Screening of Coal

256. Mine cars to be uniform in capacity.- In every bituminous coal mine in this Commonwealth where coal is mined by measurement, all cars filled by miners or their laborers shall be uniform in capacity at each mine. No unbranded cars shall enter the mine for longer than three months without being branded by the mine inspector. Violations of this section are subject to a fine of not less than $1 per car for each day the car is not in conformity with this act. This section shall not apply to mines which do not use more than ten cars. (1883, June 1, P.L. 52, sec. 2.)

257. Employment of checkweighman; settling differences as to capacity of cars and correctness of scales.- At every bituminous coal mine in this Commonwealth, where coal is mined by weight or measure, the miners, or a majority present at a meeting called for that purpose, may employ a competent person as checkweighman or check measurer, who shall be permitted at all times to be present at the weighing or measurement of coal, have power to weigh or measure the coal and, during regular working hours, to balance and examine the scales or measure the cars; this to be done so as not to interfere with the regular working of the mines. The checkweighman or measurer shall not be interfered with or intimidated by any person. He shall credit each miner with all merchantable coal mined by him on a proper sheet or book kept by him for that purpose. When differences arise between the checkweighman or check measurer and the agent or owners of the mine as to the uniformity, capacity, or correctness of scales or cars used, the dispute shall be referred to the mine inspector, whose duty it shall be to regulate the scales or cars at once. If said scales or cars prove to be correct, then the party applying for the testing shall bear all expenses thereof; if not correct, the owner of the mine shall bear such expenses. (Id., sec. 3.)

258. Weighing of coal before screening.- It shall be unlawful for any mine owner, lessee, or operator of any bituminous coal mine employing miners at bushel or ton rates, or other quantity, to pass the output of coal mined by the miners over any screen or other device which shall take any part from the weight, value, or quantity thereof, before the coal shall have been weighed and duly credited to the employee sending it to the surface to be accounted for at the legal rate of weight. Violations of this act are a misdemeanor, subject to fine and imprisonment.[9] (1897, July 15, P.L. 286.)

9 This act is unconstitutional. Commonwealth v. Brown, 8 Pennsylvania Supr. Ct. Rept. 339, 6 Pennsylvania Dist. Rept. 773, 20 Pennsylvania County Ct. Rept. 250.

Article 8. Qualifications and Duties of Particular Employees

259. Duties of miners.- The miner shall examine his working place before beginning work and take down all dangerous slate, or make it safe by properly timbering it, before commencing to mine or load coal. He shall examine his place to see whether the fire boss has left the date marks indicating his examination thereof, and if said marks can not be found he shall notify the mine foreman or his assistant. The miner shall keep his working place in a safe condition during working hours. If he at any time finds his place becoming dangerous he shall at once cease working and inform the mine foreman or his assistant of the danger, but before leaving his place he shall put some plain warning across the entrance thereto. He shall mine his coal properly before blasting, set sprags under the coal while undermining, and after each blast exercise care in examining the roof and coal and shall secure them safely before beginning to work. The miner shall order all necessary timbers at least one day in advance of needing them, as provided for in the rules of the mine. If he fails to receive the timbers and finds his place unsafe, he shall vacate it until the timbers are supplied. Under no condition shall he use coal-dust or other combustible material for tamping in any gaseous mine. When places are liable to generate sudden outbursts of explosive gas, no miner shall be allowed to charge or fire shots except under the supervision of the mine foreman, assistant mine foreman, or some other competent person designated by the foreman for the purpose. The miner shall remain during working hours in the working place assigned to him and shall not leave the place for another without permission of the foreman, assistant, or fire boss, and he shall not wander about the hauling roads or enter abandoned or idle workings. (1911, June 9, P.L. 756, Art. XXV, rule 1.)

260. Duties of driver.- When a driver has occasion to leave his trip he must see that it is left, when possible, in a safe place, secure from cars or other dangers and where it will not endanger other persons. He must take care while taking his trip down grade to have the brakes or sprags so adjusted that he can keep the cars under control, and shall not leave cars standing where they may materially obstruct the ventilating current, except in case of accident, which he shall promptly report to the mine foreman or his assistant. He shall not allow any person to ride on loaded mine cars or to drive his horses or mules. When opening a door for passing his trip through, he shall see that the door is immediately closed thereafter. (Id., rule 2.)

261. Duties of trip rider.- The trip rider shall exercise care in seeing that all hitchings are safe for use and that all the trip is coupled before starting. Should he at any time see any material defect in the rope, link, or chain he shall immediately remedy the defect, or if unable to do so shall detain the trip and report the matter to the mine foreman or his assistant. He shall not allow any person to ride on the full trip, nor on the empty trip except by the authority of the mine foreman; and the speed shall not exceed 6 miles an hour. (Id., rule 3.)

262. Duties and qualifications of hoisting engineer.- The engineer, who shall be a sober, competent person over 21 years of age, shall carefully watch over his engine and machinery under his charge and see that the steam pressure does not exceed the limit allowed by the superintendent. He shall make himself acquainted with the signal codes provided for in this act. He shall not allow any unauthorized person to enter the engine house, nor allow any person to handle or run the engine without the permission of the superintendent. When workmen are being lowered or raised, he shall take special precautions to keep the engine well under control. (Id., rule 4.)

263. Duties of motorman and locomotive engineer.- The motorman or locomotive engineer shall keep a sharp lookout ahead and sound the whistle or alarm bell frequently when approaching the parting switches or landings, and shall not exceed the speed allowed by the mine foreman. He shall see that the motors, cables, and controlling parts are kept clean and in safe operating condition, and that the headlight is burning properly when the locomotive is in motion. He shall not allow any person except his attendant to ride on the locomotive or on the full cars. (Id., rule 5.)

264. Duties of fireman.- Every fireman in charge of a boiler for the generation of steam shall keep a careful watch over it, see that the steam pressure does not exceed the limit allowed by the superintendent, frequently try the safety valve, and shall not increase the weight on the valve. He shall maintain a proper depth of water in each boiler; if anything prevent this he shall report it without delay to the superintendent or other person designated by him, and take necessary action for the protection of life and property. (Id., rule 6.)

265. Duties of fan engineer.- The engineer in charge of the ventilating fan shall keep it running at such speed as the mine foreman shall direct in writing, and shall report promptly to the mine foreman or his assistant any defect in the pressure gage and any accident to the boiler or fan machinery. If only ordinary repairs become necessary he shall await instructions of the mine foreman or his assistant before stopping the fan. Should it become impossible to run the fan or become necessary to stop it to prevent its destruction, he shall at once notify the superintendent or mine foreman, who shall give immediate warning to persons in the mine. (Id., rule 7.)

266. Duties and qualifications of furnaceman.- The furnaceman shall be over 18 years of age, shall attend to his duties with regularity, and when necessary to be off duty shall give timely notice to the mine foreman. The furnaceman shall keep a clear, brisk fire, which must not be smothered with coal or slack during working hours, and he shall not allow ashes to accumulate excessively on or under the bars or in the approaches to the furnace, and ashes shall be cooled before being removed. (Id., rule 8.)

267. Duties and qualifications of hooker-on.- The hooker-on at the bottom of any slope shall be over 18 years of age, and shall see that cars are properly coupled to a rope or chain, and that the safety catch or other device is properly attached to the rear car, before giving the signal to the engineer. He shall not allow any person to ride up the slope on the full trip other than the trip rider. (Id., rule 9.)

268. Duties and qualifications of cager.- The cager at the bottom of any shaft shall be over 18 years of age. He shall not withdraw the car until the cage comes to a rest, and when putting the full car on the cage he must see that the springs or catches are properly adjusted so as to keep the car in its proper place before he signals the engineer. (Id., rule 10.)

269. Duties and qualifications of footman.- At every shaft or slope, where persons are lowered into or hoisted from the mine a footman, who shall be over 21 years of age, shall be designated by the mine foreman. He shall be at his proper place from the time persons begin to descend until all persons quitting work at the end of the day shall be hoisted. He shall attend to the signals and see that the provisions of this act in respect to hoisting persons are complied with. The footman shall not allow tools to be placed on the same cage with men or boys, or on either cage when they are being hoisted out of the mine, except for repairing the shaft or machinery therein. The men shall place their tools in cars provided for that purpose, which cars shall be hoisted before or after the men. He shall see that no person ascends the shaft with any horse or mule unless it is secured in a suitable box or safely penned, and in any case only the driver shall accompany it. The footman shall immediately inform the mine foreman of any violation of this rule or of general rule 15 (paragraph 276). (Id., rule 11.)

270. Duties and qualifications of topman.- At every shaft or slope where persons are lowered into or hoisted from the mine, a topman or trip rider, who shall be over 21 years of age, shall be designated by the superintendent or mine foreman. His duties with respect to lowering persons, animals, and tools are similar to those prescribed for the footman in the preceding paragraph. The topman of a slope or incline plane shall be careful to close the safety block or other device as soon as the cars have reached the landing. In no case shall the device be withdrawn until the cars are coupled to the rope or chain and the proper signal given. He shall carefully inspect each day all machinery in and about the check house and the rope used for lowering the coal, shall promptly report to the superintendent any defect discovered, and shall use care in attaching securely the cars to the rope and in lowering them down the incline. He shall ring the alarm bell in case of accident, and when necessary immediately set free to act the drop logs or safety switch. He shall see that the springs or keeps for the cage to rest upon are kept in good order, and when taking the full car off he must be careful that no coal or other material is allowed to fall down the shaft. The topman shall report to the superintendent any violation of general rule 15 (paragraph 276). (Id., rule 12.)

Article 9. General Rules Concerning Conduct of or Toward Employees or Strangers

271. Unauthorized and intoxicated persons.- No unauthorized person shall enter the mine without permission from the superintendent, and no person in a state of intoxication shall be allowed to go into or loiter about the mine. (1911, June 9, P.L. 756, Art. XXV, gen. rules 1, 2.)

272. Employment of workmen; information as to rules.- No person shall be employed to blast coal, rock, or slate unless the mine foreman is satisfied that he is qualified by experience to perform the work with ordinary care, and no inexperienced person shall be employed to mine out pillars unless in company with one or more experienced miners. Every workman, when first employed, shall have his attention directed by the mine foreman or his assistant to the general and special rules contained in this act. Said rules shall be posted in a conspicuous place at or near the main entrance to the mine, and shall be printed in the various languages of the employees. (Id., rules 3, 4, 6.)

273. Examination of working place; reporting unsafe conditions.- Every workman in the mine shall examine his working place before commencing work and after any stoppage of work during the shift. All employees shall notify the mine foreman or his assistant of the unsafe condition of any working place, hauling roads, or traveling ways, or of damage to doors, brattices, or stoppings, or of obstructions in their passages, when said conditions are known to them. (Id., rules 5, 7.)

274. Traveling or riding on slopes, planes, etc.- No person shall be allowed to travel on foot to and from his work on any hoisting slope, incline plane, dilly, or locomotive road, unless no other roads are provided for that purpose. No person shall ride upon or against any loaded car or cage in any shaft or slope, and except as otherwise provided in this act, no person other than the trip rider shall be permitted to ride on empty trips on any slope, incline plane, or dilly road; and no person, other than the driver or trip rider, shall be allowed to ride on the full cars. (Id., rules 8-10.)

275. Tampering with electric wires, danger signals, etc.- Any person defacing or pulling down any notice board, danger signal, general or special rules, record books, or mining laws, shall be prosecuted by the superintendent. Persons are forbidden to meddle or tamper in any way with any electric or signal wires, or any other equipment in or about the mine. (Id., rules 11, 12.)

276. Number of persons to be hoisted; duty to furnish empty cage or car.- No greater number of persons shall be lowered or hoisted at any one time, in any shaft or slope, than is permitted by the inspector. Whenever the said number of persons returning from work shall arrive at the bottom of the shaft or slope, in which persons are regularly hoisted or lowered, they shall be promptly furnished with an empty cage or car and be hoisted. In cases of emergency a less number than the permitted number shall be promptly hoisted.

Persons crowding or pushing to get on or off the cage or car shall be deemed guilty of a misdemeanor, and the superintendent shall discharge or prosecute them when the matter is reported to him by the topman or footman. (Id., rule 15.)

277. Boreholes when cutting faults, etc., in gaseous mines.- In cutting clay veins, spars, or faults in entries or other narrow workings, going into the solid coal, in mines wherein explosive gas is being generated in dangerous quantities, a borehole shall be kept not less than 3 feet in advance of the face of the work, or 3 feet in advance of any shot hole drilled for a blast to be fired in. (1915, May 13, P.L. 310.)

278. Removal of gas by brushing; ignition of gas.- An accumulation of gas shall not be removed by brushing, or when persons in the mine may be endangered thereby. When gas is ignited by a blast, or otherwise, the person having charge of the place shall immediately extinguish it, if possible, and if unable to do so he shall immediately notify the mine foreman or his assistant. Miners must see that no gas blowers are left burning upon leaving their working places. It shall be the duty of the superintendent to notify the mine inspector of any violation of this rule, and the inspector shall then prosecute the offender. (1911, June 9, P.L. 756, Art. XXV, gen. rules 19, 20.)

279. Fencing of abandoned places; not to be entered.- Every abandoned slope, shaft, air hole, or drift shall when so abandoned be properly fenced around or across its whole entrance. No person shall go into an old or abandoned portion of a mine, or into any other place not in actual course of working, without permission from the mine foreman, and no person shall travel to and from his work except by the traveling way assigned for that purpose. The mine foreman shall prosecute all persons violating this rule. (Id., rules 26, 27.)

280. Interference with or pollution of air prohibited.- No person shall commit any nuisances, or thow into, deposit, or leave coal or dirt, stones, or other rubbish, in the airway or road, so as to interfere with, pollute, or hinder the air passing into and through the mine. (Id., rule 28.)

281. Drainage into mine or entry prohibited.- If any person shall cause or permit to be constructed or used, any sewer or other method of drainage from any building for the carrying of sewage, offal, refuse, or other offensive matter, into any operating mine, or any entry way, passage, or room in any mine (such entry way, passage, or room being used for ventilating or drainage purposes, or for a traveling way), such person shall be guilty of a misdemeanor, punishable by fine not exceeding $1,000 and imprisonment not exceeding one year, or both. (1911, June 9, P.L. 756, Art. IX, sec. 10.)

282. Steam pipes to be encased.- No steam pipes through which high-pressure steam is conveyed for driving pumps or other machinery shall be laid on traveling or haulage ways, unless encased in asbestos or other nonconducting material, or so placed that the radiation of heat into the atmosphere will be prevented as far as possible. (1911, June 9, P.L. 756, Art. XXV, gen. rule 29.)

283. Ventilation where steam locomotives are used.- When a steam locomotive is used for hauling coal out of a mine, the tunnels through which it passes shall be properly ventilated and kept free as far as practicable of noxious gases, and a ventilating apparatus shall be specially provided by the operator to produce such ventilation. (Id., rule 30.)

284. Code of signals.- In all shafts and slopes where persons, coal, and other materials are hoisted by machinery, the following code of signals shall be used;

One rap or whistle - to hoist coal.
One rap or whistle - to stop car or cage when in motion.
Two raps or whistles - to lower car or cage.
Three raps or whistles - to hoist persons. The engineer shall signal back when ready, after which the person shall get on the car or cage, and then one rap or whistle shall be given to hoist.
Four raps or whistles - to turn on steam to the pumps. (Id., rule 31.)

285. Carrying matches and smokers' articles into mines.- No person shall carry any matches, pipes, or other smokers' articles into a mine, or portion thereof, worked exclusively with locked safety lamps, nor shall he have any such articles in his possession while in such a mine. (Id., rule 32.)

286. Rules, books, etc., to be printed and furnished to operators.- The special and general rules in the various languages and all books, blank forms, and notices mentioned in this act, shall be printed by the State, and shall be furnished to the operators by the department of mines, through the inspectors, and all record books shall at all times be accessible to inspectors. The operators shall pay to the department of mines the actual cost to the State of the assistant mine foreman's report book, the mine foreman's daily record book, the mine foreman's weekly record book, and the fire boss's daily record book. (Id., rule 34; amended, 1923, May 28, P.L. 457, sec. 1.)

Article 10. Equipment of Mines; Boilers; Inside Stables

287. Signaling and hoisting apparatus; safety catches; handrails; etc.- The operator or superintendent shall provide and maintain from the top to the bottom of every shaft or slope where persons or material are lowered or hoisted, a telephone or metal tube suitably adapted to the passage of sound, through which conversation may be held between persons at the top and bottom of the shaft or slope, and shall also provide means of signaling between those points. The same provision shall apply to inside planes whereon coal is lowered and persons have to travel, when required by the inspector. In all gaseous mines telephone connection shall be made from the surface to the main section of the mine. Signaling apparatus and telephone connections shall be kept in good condition and always available for service. The operator or superintendent shall provide every cage, used for lowering or hoisting persons, with handrails at sides or overhead, with chain, bar, or gate at ends, and with a sufficient covering overhead to protect persons thereon. Each cage

shall be provided with efficient safety catches, which shall be tested every two months, and a record of each test sent to the inspector and the superintendent and recorded in a book kept at the mine office for that purpose. The ropes shall be securely attached to the sides of the drum of every machine used for lowering and hoisting persons or material, and the flanges shall have a clearance of at least 4 inches when all the rope is wound on the drum. Adequate brakes shall be attached to the drum so that the speed can be controlled. An efficient indicator showing the position of the cages in the shaft shall be attached to the hoisting apparatus, and a safety device that will prevent overwinding shall be attached to every engine used for lowering and hoisting persons. All shafts shall be provided with safety gates, approved by the inspector, controlled by the cage at the top and intermediate landings. (1911, June 9, P.L. 756, Art. VIII, sec. 1.)

288. Ropes, links, chains; inspection.- The main coupling chain attached to the socket of the wire rope of every shaft shall be of the best-quality iron and tested to the satisfaction of the inspector, the manner and result of testing to be entered in a book. Bridle chains of the same quality of iron shall be attached to the main hoisting rope 3 feet above the socket from the top crosspiece of the cage. In shafts where coal is hoisted and employees lowered into or hoisted from the mine, the ropes, links, and chains shall be of ample strength, with a factor of safety of not less than 5 to 1 of the maximum load. In shafts used exclusively for lowering or hoisting employees and material the factor of safety or ropes, links, and chains shall be at least 10 to 1 of the maximum load. All such ropes, links, and chains shall be examined at least once every 24 hours by a competent person delegated for that purpose by the superintendent. Any defect therein found by which life and limb may be endangered shall be reported at once to the superintendent, who shall immediately proceed to remedy the defect, and until doing so no person shall be lowered or hoisted by the defective apparatus. The person making said examination shall make a daily record of each inspection in a book kept at the mine office for that purpose, and shall send a copy each day to the superintendent. (Id., secs. 2, 3.)

289. Guard railings around machinery.- All machinery in and about the mines from which an accident would be liable to occur shall be properly fenced off by suitable guard railing. (Id., sec. 4.)

290. Washhouses, when required; equipment.- When the clothing of employees in any mine becomes wet from working in wet places therein, the operator or superintendent, at the request of the inspector who shall make such request upon petition of any 10 employees working in the aforesaid wet places, shall provide a suitable building, convenient to the principal entrance of the mine, for the use of persons employed in wet places therein for washing themselves and changing their clothes. The building shall be maintained in good order, properly lighted and heated, and provided with hot and cold water and facilities for persons to wash. Any operator, superintendent, or inspector failing to comply with this provision, or any person maliciously injuring the building or any of its appliances, shall be guilty of a misdemeanor. (Id., Art. XIV, sec. 1.)

291. Inspection of steam boilers; safety valves.- All steam boilers in and about the mines shall be kept in good condition. The superintendent shall have them inspected by a duly qualified person every six months, and the report of such inspection shall be posted at the mine office. Each boiler shall be provided with a safety valve of sufficient area for the steam to escape, and with weights or springs properly adjusted. (Id., Art. VIII, secs. 6, 7.)

292. Consent of inspector for boiler inside mine.- No boiler used for generating steam shall be allowed inside of any mine without the written consent of the inspector. If the inspector consents the boiler shall be enclosed in a fireproof building within 50 feet of the bottom of an upcast shaft, which shall not be less than 35 square feet in area. (Id., sec. 8.)

293. Steam gages.- Every boiler house shall be provided with a sufficient number of properly connected steam gages to indicate the steam pressure to the firemen, outside foreman, or superintendent; and another steam gage shall be attached to the main steam pipe in the engine house so that the hoisting engineer can readily examine it. (Id., sec. 9.)

294. Construction of inside stables; storage of hay and straw.- The superintendent or mine foreman shall not provide a horse or mule stable inside of any mine unless space for it is excavated in solid strata of rock, slate, or coal. If excavated in the coal seam, the wall shall be of brick, stone or concrete not less than 12 inches thick, and shall be built from the bottom slate to the roof. Wood or other combustible material shall be used in the smallest practicable quantity in the construction of the inside of the stable. The air current used for ventilating the stable shall not be intermixed with air used for ventilating other portions of the mine, but shall be conveyed directly to the return air current. No open light shall be permitted in any inside stable. No hay or straw shall be taken into any mine unless pressed and made into compact bales, which shall be kept in a storehouse built apart from the stable and in the same manner as the stable. Under no circumstances shall the hay be stored in the stable. (Id., Art. XV, sec. 1.)

Article 11. Openings and Outlets; Sinking of Shafts

295. Two openings required; exceptions.- It shall not be lawful to employ any person to work in a mine unless there are at least two openings or outlets to the surface from every seam of coal being worked, and available from every entry thereof, which shall have distinct means of ingress and egress available at all times for the use of employees. The distance between two shafts shall not be less than 200 feet, between openings to the surface of slopes not less than 150 feet, and between drifts not less than 50 feet. The distances specified may be less with the consent of the inspector, and they apply only to mines opened after the passage of this act. The passageways between two shafts shall be maintained in safe condition for employees to travel therein and the pillars in entries between such shafts shall not be removed without the consent of the inspector. The foregoing requirements shall not apply to the openings of a new mine or a new entry of a mine that is being worked for

making connection between the two outlets as long as not more than 20 persons are employed at such work at any one time, nor to any mine in which the second opening has been rendered unavailable by the final robbing or removing of pillars, as long as not more than 20 persons are employed therein at any one time. (1911, June 9, P.L. 756, Art. VI, sec. 1.)

296. Cages, etc., in mines having only one outlet.- The cages or other safe means of egress shall be always available for the persons employed in any mine that has no second outlet available. (Id., sec. 2.)

297. Passageway around shaft.- There shall be around the side of every hoisting shaft, at the bottom and at intermediate points where it intersects any entry, a passageway 5 feet high and 3 feet wide in the clear. The passageway shall be cut through the solid strata or constructed of masonry, and shall be kept open at all times so as to enable persons to pass around the shaft. (Id., sec. 3.)

298. Entries to be provided; traveling ways.- Every mine generating explosive gas opened after the passage of this act shall have at least four main entries, two leading from the main opening and two from the second opening into the body of the mine. In such mines where locked safety lamps are used exclusively, projected to open up a large acreage with main entries 5000 feet or more in length, shall have at least five main entries, two of which shall lead from the main opening and two from the second opening, and the fifth (which may be connected with an opening to the surface or with the intake airway at or near the main intake opening) shall be used exclusively as a traveling way for employees.

Every nongaseous mine opened after the passage of this act shall have at least two main entries, one leading from the main opening and one from the second opening into the body of the mine. In every new nongaseous mine projected to open up a large acreage with main entries 5000 feet or more in length, the operator shall either haul the employees into and out of the mine at the beginning and end of each shift or provide at least three main entries, one leading from the main opening and one from the second opening, and one (which may be connected with an opening to the surface or with the intake airway at or near the main intake opening) shall be used exclusively as a traveling way.

In any mine opened as a nongaseous mine which afterwards becomes a gaseous mine, and in gaseous mines opened prior to the passage of this act, where locked safety lamps are used exclusively, having less than five main entries that have reached 5000 feet or more in length, and are to be extended 2000 feet or more, the superintendent shall have a new opening of ample dimensions made from the surface, if the inspector of the district and two additional inspectors appointed by the secretary of mines shall deem such additional opening necessary. The main entries and the traveling way shall be extended from this opening to the face of the workings. The operator may continue to work the mine under the requirements of this act for nongaseous mines until by due diligence he can change conditions to meet the requirements of this paragraph.

The intake and return entries shall be kept drained and free from obstructions so that persons may safely travel therein, and have a safe means of egress from workings in case of emergency. Such entries shall be separated by pillars of coal of sufficient strength and shall not be driven more than 200 feet beyond the last cut-through, except for exploratory purposes. When the main entry of a nongaseous mine, or both main entries of a gaseous mine, used for intake for air, are also used for mechanical haulage, a separate traveling way, leading into the body of the mine shall be provided for the use of employees, or they shall be hauled into and out of the mine at the beginning and end of each shift.

In all mines where the coal seam is less than $3\frac{1}{2}$ feet high such traveling way shall be at least $4\frac{1}{2}$ feet in height; in mines where the seam is 4 feet high such traveling way shall be at least 5 feet in height; and the width shall not be less than 6 feet. All such traveling ways shall be well drained, and kept free from refuse, smoke, noxious gases, and electric wires, unless said wires are so placed and maintained as not to endanger life. (Id., sec. 4.)

299. Overcast and undercast.- In every slope with workings on both sides an overcast or an undercast, not less than 5 feet wide and 5 feet high shall be provided as a passageway for the use of employees to cross from one side of the slope to the other. The overcast or undercast shall connect with available passageways leading to the workings on both sides of the slope. The intervening strata shall be of sufficient strength at all points to insure safety to the employees. If impracticable to drive it in the solid, an overcast or an undercast if substantially built with masonry or other incombustible material will be deemed sufficient. (Id., sec. 5.)

300. Shaft less than 100 feet deep; stairways.- In mines opened after the passage of this act, if the opening or outlet other than the main opening is a shaft not more than 100 feet deep and is used by employees for ingress or egress it shall be kept available and in safe condition, free from steam, dangerous gases and obstructions. It shall be fitted with safe and convenient stairways, made safe by having handrails of suitable material placed on one side, or on both sides when required by the inspector, and shall be inspected every 24 hours by a competent person employed for that purpose. Water that may come from the surface or the strata in the shaft shall be conducted away so that it will not fall on the stairways or on persons while using them. (Id., sec. 6.)

301. Shaft more than 100 feet deep.- After passage of this act, when a mine is operated by a shaft more than 100 feet deep the persons employed therein shall be lowered and hoisted by means of machinery unless the second opening is a drift or a slope. When the employees are lowered into or hoisted from the mine at the main shaft opening the second opening, if a shaft, shall also be supplied with a stairway, as provided in the preceding section, or with suitable machinery for safely lowering and hoisting persons in case of emergency. (Id., sec. 7.)

302. Angle of descent in slope; lowering and hoisting.- At any mine where one of the openings hereinbefore required is a slope and is used as a means of ingress and egress by the employees, and where the angle of descent of the slope exceeds 15° and its length from the mouth of the opening exceeds 1000 feet, or the angle of descent averages from 5 to 15° and its length exceeds 3000 feet, the employees shall be lowered into and hoisted from the mine at the beginning and end of each shift at a speed not to exceed 6 miles per hour. When a separate traveling way is provided, the owner, at his option, is exempt from the requirements of this section if the angle of the traveling way does not exceed 20°. (Id., sec. 8.)

303. Number of persons to be lowered or hoisted; speed of cage.- No greater number of persons shall be lowered or hoisted at any one time than permitted by the inspector; and notice, signed by the inspector, of the number so allowed shall be kept posted by the operator or superintendent in conspicuous places at the top and bottom of the shaft. The speed of the cage, when lowering or hoisting persons shall not exceed 900 feet a minute. (Id., Art. VIII, sec. 5.)

304. Driving drifts across property lines for removal of accumulated water.- In any mine wherein water may have been allowed to accumulate in dangerous quantities, endangering adjacent mines and miners working therein, and when such can be tapped and set free and flow by its own gravity to any point of drainage, it shall be lawful for any operator having mines so endangered, with the approval of the inspector of the district, to proceed to remove the danger by driving a drift and to drive across property lines if needful. It shall be unlawful for any person to dam or obstruct the flow of any stream from the mine when so set free, on any part of its passage to point of drainage. (Id., Art. XII, sec. 3.)

305. Condemnation of adjoining lands.- Sections 1 and 4 of Article XII of the act of June 9, 1911, provides for the condemnation of adjoining land where otherwise impracticable to drain or ventilate a mine or to comply with requirements of the law as to ways of ingress and egress. However, these sections have been held unconstitutional and are omitted here. (Poland Coal Co.'s Case, 58 Pa. Super. Ct. Rep. 312.)

306. Sinking of shafts; structures at top.- There shall be erected over every shaft being sunk a safe and substantial structure to sustain sheaves or pulleys, ropes and loads, at a height of not less than 20 feet above the tipping place. The top of such shaft and landing platform shall be arranged so that no material can fall into the shaft while the bucket is being emptied. The structure shall be erected as soon as substantial foundation is obtained, and in no case shall a shaft be sunk more than 50 feet without such structure. If provision is made to land the bucket on a truck, the truck and platform shall be so constructed that material can not fall into the shaft. (Id., Art. VII, secs. 1, 2.)

307. Id.; bucket or cage; drum; guides.- Rock and coal from shafts being sunk shall not be raised except in a bucket or on a cage, which must be connected with the rope by a safety hook, clevis, or other safe attachment. The rope shall be fastened to the side of the drum and not less than three coils of rope shall always remain thereon. If the shafts are 100 feet or more in depth, they shall be provided with guides and guide attachments, applied so as to prevent the bucket from swinging while being lowered or hoisted; and the guides and attachments shall be maintained not more than 75 feet from the bottom of the shaft. (Id., sec. 3.)

308. Id.; sides of shaft; gas; blasting; ventilation.- It shall be the duty of the person in charge of shaft sinking to see that the sides of all shafts are properly secured for safety, and that no loose rock or material is allowed to remain on any timber on top or in the shaft after each blast. Where explosive gas is encountered he shall see that the shaft is examined before each shift and before the men descend after each blast, and also that the place is safe. (Id., sec. 4.)

309. Id.; brake; inspection of equipment; hoisting and lowering persons.- An efficient brake shall be attached to every drum of an engine used for sinking shafts, and all machinery, ropes, and chains connected therewith shall be examined once every 12 hours. Not more than four persons shall be lowered or hoisted in or on a bucket in any shaft at one time; and no person shall ride on a loaded bucket. (Id., secs. 5, 6.)

Article 12. Ventilation and Preventing Accumulation of Gases

310. Quantity of air; mode of ventilation; permanent doors forbidden.- The operator or superintendent of every mine shall maintain ample means of ventilation to furnish a constant and adequate supply of pure air for employees. The minimum quantity of air for a nongaseous mine and one generating explosive gas shall be, respectively, 150 and 200 cubic feet per minute per person employed therein, and as much more as one or more of the inspectors may deem requisite. The return air from each split where from 70 to 90 persons are employed shall be conducted by an overcast or undercast into the return airway, which shall lead to the main outlet. The ventilation shall be conducted through the main entries, cross entries, and to the working faces of all working places in the mine in sufficient quantities to dilute, carry off and render harmless smoke and gases generated therein to such an extent that all working places and traveling roads shall be in a safe and healthful condition. No permanent door shall be allowed in the main entry unless its removal shall be deemed impracticable by the inspector. (1911, June 9, P.L. 756, Art. IX, sec. 1.)

311. Furnace ventilation; number of persons in one air current.- Where five or more persons are employed at any one time in a mine, the operator or superintendent shall provide ample ventilation in accordance with the preceding section. It shall not be lawful to use a furnace for ventilating any mine wherein explosive gas is being generated. Not more than 70 persons shall be

permitted to work in the same continuous air current unless in the judgment of the inspector it is impracticable to comply with this requirement, in which case a larger number, not exceeding 90, may be permitted to work therein. (Id., sec. 2.)

312. Cut-throughs; regulations when room and pillar systems are not used.- Cut-throughs in entry pillars and in pillars of rooms driven in the "room and pillar" system of mining shall be provided not less than 16 yards nor more than 35 yards apart. In mines or portions thereof developed for mining by a system other than the "room and pillar," all openings except entries may be driven 100 yards without cut-throughs, provided the following regulations are enforced:

1. That sufficient air be circulated along the face of each entry, cut-through chamber, or other opening to sweep away and render harmless all smoke, and noxious or explosive gases.

2. In gaseous mines there shall be kept at the face of every working place, while the men are at work, at least one approved flame safety lamp, if such place is driven more than 105 feet without a cut-through.

3. That in every mine where a working place is driven more than 105 feet without a cut-through, said place shall be examined by a mine official at least three times a day while the men are or should be at work.

4. In gaseous mines where it is necessary to drive openings more than 105 feet off any entry or other road, not more than four such places shall be advanced at the same time, and not more than six places shall be advanced at the same time in any air split without proper connection with the air current.

5. Booster and/or blower fans shall not be used in gaseous mines for ventilating workings having no connection with the air circuit unless equipped with Government-approved, flameproof, electric motor; and the location of such fans shall have the approval of the inspector.

6. In all gaseous mines where places are driven more than 105 feet without the formation of an air circuit, the coal-dust in the entries shall be rendered inert to explosibility by the application of shale-dust or other incombustible material, and the coal-dust in all other openings shall be taken care of as provided by law. (1927, Apr. 27, P.L. 390, sec. 1.)

313. Measurement of air.- The quantity of air passing a given point shall be ascertained by an anemometer, the measurements to be taken weekly by the mine foreman, at or near the main inlet and outlet airway in the mine, and also at the last cut-through in the last room and in the entry beyond the last room turned. Such measurements shall be taken on days when the men are at work. (1911, June 9, P.L. 756, Art. IX, sec. 4.)

314. Stoppings; construction.- All new stoppings in cut-throughs between the main intake and return airways shall be substantially built of masonry, concrete, or other incombustible material, and in mines generating explosive gas all new stoppings and renewals of old stoppings in cross entries shall be built of such material. Stoppings in cross entries in nongaseous mines may be built of timber. All stoppings shall be kept in good condition so as to keep the air up to the working faces. Temporary stoppings shall be erected in cut-throughs in rooms to conduct the ventilation to the face of each room, and such stoppings may be constructed of timber or brattice cloth. (Id., sec. 5.)

315. Fans; operation; recording instruments; ventilating furnaces.- Ventilating fans shall be kept in operation continuously day and night unless operations are definitely suspended, except, in the case of nongaseous mines, when written permission is given by the inspector to stop it. A copy of such permission shall be posted at the entrance to the mine and shall state the hours during which the fan may be stopped. The inspector shall have the power to withdraw or modify such permission at any time. In all cases the fan shall be started two hours before the time to begin work. Should it become necessary to stop the fan at any mine on account of accident or other unavoidable cause, it shall be the duty of the mine foreman or his assistant in charge, after first having provided for the safety of the persons employed in the mine to order the fan stopped for necessary repairs. Every ventilating fan shall be provided with a recording instrument by which the revolutions or the effective ventilating pressure of the fan shall be registered. The registration for each day shall be kept in the office at the mine for one year.

No principal ventilating fan shall be placed inside of any mine, nor shall an auxiliary fan be so placed unless driven by electricity or compressed air. If the fan be electrically driven the motor shall be placed in the intake airway. Every ventilating furnace in a mine shall be properly attended to and operated by a competent person, employed by the mine foreman for that purpose, for two hours before the appointed time to begin work and constantly thereafter during working hours. (Id., sec. 6.)

316. Accident to ventilating fan.- In case of accident to a ventilating fan or its machinery, whereby the ventilation would be seriously interrupted, the mine foreman shall order the men to withdraw immediately from the mine and shall not allow them to return until the ventilation has been restored and the mine thoroughly examined and reported safe. (Id., Art. IV, sec. 4.)

317. Air bridges, overcasts, etc.- All new air bridges, overcasts, or undercasts shall be substantially built of masonry, concrete, or other incombustible material or shall be driven through the solid strata. The mine foreman shall see that these bridges are properly built and are of ample strength. (Id., Art. IX, sec. 7.)

318. Ventilating doors.- In every mine the doors used for guiding and directing the ventilation shall be so hung and adjusted that they will close of themselves, or shall be supplied with springs or pulleys so that they can not remain open. All principal doors shall be so placed that when one door is open another which has the same effect upon the same current shall be and remain closed to prevent stoppage of the air current. An attendant shall be employed at each principal door, controlling the main air current in the entries, through which cars are hauled, for opening and closing it, unless a self-acting door, approved by the inspector, is used. A shelter hole shall be provided at each door to protect the attendant from danger from cars. Attendants shall remain at the doors during working hours, but the same attendant may attend two doors if his absence from the first does not endanger the safety of the employees. At every door on any inclined plane or road whereon haulage is done by machinery an attendant shall always be on duty during working hours. At every door on the plane or road and wherever a principal door is placed, an extra door shall be provided for use in case of necessity. (Id., sec. 8.)

319. Petroleum, alcohol, etc., as motive power, prohibited.- No product of petroleum or alcohol, or any compound that in the opinion of the inspector will contaminate the air to such an extent as to be injurious to health, shall be used as motive power in any mine. (Id., sec. 9.)

320. Traveling in return air current of gaseous mine.- The employees of a gaseous mine, or any portion thereof, are prohibited from traveling into or out of the mine in the return air current, if explosive gas can be detected by an approved safety lamp in the air current. (Id., Art. X, sec. 5.)

321. Air current to be measured weekly.- The mine foreman or his assistant shall, at least once a week, on days when the men are at work, measure the air current at or near the main inlet and outlet airway and in the last cut-through in the last room and in the entry beyond the last room turned in each entry, and make a record of same, as provided in section 18 of this article (paragraph 242). For making said measurements an anemometer shall be provided and kept in good condition by the superintendent. (Id., Art. IV, sec. 3.)

322. Notice of danger; report to inspector; stoppings.- The mine foreman shall notify the superintendent in writing whenever in his opinion the mine is becoming dangerous through the lack of ample ventilation, resulting in the accumulation of gas or coal-dust in various portions of the mine. The superintendent shall then notify the inspector, requesting him to make a personal examination. If the inspector finds it is becoming dangerous he shall at once direct the superintendent to have it put in safe condition, and if necessary, have an additional opening of ample dimensions sunk from the surface to the interior to be used as an outlet or inlet for air and also as an escapeway. In mines generating explosive gas in quantities sufficient to be detected by an approved safety lamp the mine foreman shall see that, when the permanent station of the fire boss is located a mile or more from the mine entrance, all abandoned, finished or unfinished workings, in the intervening distance between the station and entrance are completely shut off from the main intake or manway headings by stoppings of masonry, concrete, or other incombustible material of sufficient thickness to keep the explosive or noxious gases from coming in contact with the intake air or with the persons employed therein. (Id., sec. 5.)

Article 13. Safety Lamps, Open Lights, and Electric Lamps

323. Open lights prohibited in certain places; order for locked safety lamps; appeal; commission.- The use of open lights is prohibited in any entry, airway, traveling way, room, or any other working place where explosive gas is being generated in such quantities as can be detected by an approved safety lamp, also in pillar workings where a sudden inflow of explosive gas is likely to be encountered, and all such places shall be worked exclusively with locked safety lamps. This does not prohibit the use of approved electric lamps, provided the mine foreman, assistant mine foreman, fire bosses, machine runners, shot firers, pumpers, and other persons required by the foreman, shall in addition use approved flame safety lamps for detecting explosive gas. The use of

open lights is also prohibited in all working places or other portions of the mine through which explosive gas might be carried in the air current in quantities indicating danger. If the inspector is of the opinion that any mine, or portion thereof, should be operated by the use of locked safety lamps exclusively, he may petition the secretary of mines, setting forth such opinion and his reasons therefor; whereupon the secretary shall instruct two or more other inspectors to accompany the district inspector to make a further examination. The committee of inspectors shall within seven days make a report to the secretary and to the superintendent or operator of the mine. Their decision shall be final unless an appeal therefrom is taken within seven days to the court of quarter sessions. Whereupon the court shall appoint four competent persons, who in turn appoint a fifth. The five persons so named constitute a commission to investigate and report on the matter in dispute. The report of the commission is final unless exceptions thereto be filed within seven days of the filing of its report, in which case the court shall at once hear and determine the cause. An appeal may be taken to the supreme court. (1925, Apr. 7, P.L. 175, sec. 3.)

324. Open lights in return air currents prohibited.- The use of open lights is prohibited in the return air current of any portion of a mine ventilated by the same continuous air current that ventilates any other portion in which locked safety lamps or electric lamps are used; but this section shall not apply to a mine wherein explosive gas is generated only at the face of active entries. (Id., sec. 4.)

325. Open lights in portion of mine.- If one portion of a mine is worked by the use of locked safety lamps while another is worked by open lights, the return air from the gaseous portion shall be conducted directly into a return airway leading to the fan or outlet. When a portion of a mine is worked by locked safety lamps and other portions by open lights, the mine foreman shall provide a suitable danger station with an attendant, whose duty it shall be to see that the employees from the open-light portion do not enter the locked safety lamp portion unless they are provided with such lamps. (1911, June 9, P.L. 756, Art. X, sec. 4.)

326. Lamp station; construction of safety lamps.- When safety lamps are used the position of the lamp station for lighting or relighting shall not be in the return air current. When safety lamps are used in a mine by fire bosses or other persons, they shall be so constructed that they may be safely carried against the air current ordinarily prevailing in that portion of the mine in which the lamps are being used. (Id., secs. 6, 8.)

327. Ownership and care of safety lamps.- All safety lamps used for examining mines or for working therein shall be the property of the operator, and shall be in the care of the mine foreman, his assistant, fire boss, or a competent person appointed by the foreman whose duty it shall be to clean, fill, trim, examine, light, and deliver them locked and in a safe condition to the men when entering the mine and to receive the lamps from the men when returning from work,

for which services a charge not exceeding the actual cost of labor and material may be made by the operator. At any mine wherein explosive gas was generated within one year before the passage of this act, in sufficient quantities to be detected by an approved safety lamp, a sufficient number of safety lamps, not less than one-fourth of the number in use, shall be kept for use in case of emergency. Every person who knows that his safety lamp is defective shall return it immediately and report the fact to the person authorized to care for the lamp, who shall report the matter to the mine foreman, his assistant, or fire boss as soon as practicable. (Id., sec. 9.)

328. Persons to be entrusted with lamps; possession of keys.- No safety lamp shall be entrusted to any person, for use in a mine, until such person has given satisfactory evidence to the mine foreman that he understands the proper use thereof and the danger of tampering with the lamp. No one except a person authorized by the mine foreman shall have in his possession a key or other instrument for unlocking any safety lamp in any mine where locked safety lamps are used. Persons other than those authorized having such keys or instruments in their possession shall be prosecuted by the superintendent. (Id., Art. XXV, gen. rules 16, 17.)

Article 14. Use of Oils

329. Oiling or greasing cars; amount of lubricating oil permitted.- Oiling or greasing cars inside of any mine is prohibited unless the place where the oil or grease is used is thoroughly cleaned at least once every day. Not more than one barrel of lubricating oil shall be permitted in any mine at one time, and it shall be kept in a fireproof building, cut out of solid rock, or made of masonry or concrete of sufficient thickness to insure safety in case of fire. (1911, June 9, P.L. 756, Art. XVII, sec. 1.)

330. Oil used for illuminating purposes; storage in mine.- No explosive oil shall be used in any mine for lighting purposes, except in safety lamps, and shall not be taken into or stored in a mine in quantities exceeding 5 gallons. Oil when stored in a mine shall be kept in a fireproof vault made of masonry or concrete. Any oil or material used in open lamps shall be nonexplosive, free from odors and fumes deleterious to health, shall have a burning point not lower than 300°, and must not produce over eleven one-hundredth of 1 per cent of its weight of soot when burned in a miner's lamp with a flame $1\frac{1}{2}$ inches high; the determination of the percentage of soot to be by tests specified by the department of mines. (Id., secs. 2, 3.)

331. Paraffin wax; branding of illuminants.- Paraffin wax used in mines shall not contain over 3 per cent of oil. All illuminants sold to be used in open lamps in mines shall have branded conspicuously on the barrels or packages containing them the name of the manufacturer, date of shipment, and percentage of soot. (Id., secs. 4, 5.)

332. Use of prohibited articles; duty of inspector.- Persons selling for use or using, or any mine foreman permitting to be used, in any mine, any oil or other material for illuminating purposes other than prescribed in this article, shall be guilty of misdemeanor. Any illuminant that is found not detrimental to health and safety after proper tests can be used with the consent of the inspector. Whenever an inspector has reason to believe that an illuminant is being used, sold, or offered for sale, in violation of this article, he shall take samples of the illuminant and have them tested under the direction of the department of mines. (Id., secs. 6, 7.)

Article 15. Explosives and Blasting Operations

333. Hauling and storing explosives; amount allowed; receptacles.- No powder or high explosive shall be stored in any mine and no more of either shall be taken into any mine at one time by any one person than is required in one shift. The quantity shall not exceed 5 pounds, except that in a mine employing shot firers they may take a sufficient quantity to complete their work. Black powder shall be put up in 5, 10, 15, and 25 pound metallic cans or cannisters, or receptacles of equally safe material, and all powder shall be carried into the mine in such receptacles. No explosive shall be stored in any tipple or weighing office, and no naked lights shall be used while the attendant is weighing and giving out explosives. No black powder, high explosives, or detonators shall be hauled on any electric motor trip unless the same are encased in nonconductive boxes or receptacles. (1911, June 9, P.L. 756, Art. XVI, sec. 1; id., Art. XXV, gen. rules 13, 14.)

334. "Permissible" explosives to be used in gaseous mines.- In such portions of dry and dusty mines wherein explosive gas is being generated in quantities sufficient to be detected by an approved safety lamp, no explosives shall be used except "permissible" explosives, as designated by the testing station of the Federal Bureau of Mines. Each charge shall consist of only one kind of explosive. The department of mines shall forward to the operators, upon application, the names of all explosives on the permissible list. No "permissible" explosive shall be sold for use in bituminous mines unless the name of the manufacturer and explosive, method of handling, and full instruction for use are conspicuously displayed on or in the package containing the explosive. (Id., Art. XVI, sec. 2.)

335. Where detonators are to be kept; deterioration from permissible explosive standard.- Detonators shall be kept in securely locked cases, separate from other explosives, until required for use. The secretary of mines, when satisfied by tests that any permissible explosive has deteriorated from the standard established by the Federal Bureau of Mines, thereby becoming dangerous, may prohibit the use thereof, either absolutely or subject to conditions. (Id., sec. 3.)

336. Warning to be given before firing blasts.- When a miner or shot firer is about to fire a blast he shall notify all persons who may be endangered and shall give sufficient alarm so that persons approaching may be warned. (Id., Art. XXV, gen. rule 21.)

337. Precautions in handling explosives.- Whenever a miner or shot firer shall open a box containing explosives, or while handling them, he shall first place his lamp not less than 5 feet from such explosive and in such a position that the air current can not convey sparks to it, and he shall not smoke while handling explosives. (Id., gen. rule 22.)

338. Charging and tamping holes.- In charging and tamping a hole for blasting, no person shall use any iron or steel needle. The charger or tamping bar shall be of wood or tipped with copper. No explosive shall be forcibly pressed into a hole that is of insufficient size. When a hole has been charged the explosive shall not be taken out, and no hole shall be bored for blasting less than 12 inches from any hole when the charge has misfired. In all gaseous and in all dry and dusty mines shot firers or other persons charging holes for blasting shall use incombustible material for tamping. All holes in any mine before being fired shall be solidly tamped the full length of the hole; provided, however, with the consent of the mine inspector "cushion" or "air" blasting shall be permitted. (Id., gen. rules 23-25; amended, 1929, Apr. 30, P.L. 880, sec. 1.)

Article 16. Electricity and Electrical Apparatus

339. Rules observed when practicable.- The following rules shall be observed as far as is reasonably practicable in the mines. (1911, June 9, P.L. 756, Art. XI.)

340. Definitions.- The terms "potential" and "voltage" are synonymous and mean electrical pressure.

"Difference of potential" means the difference of electrical pressure existing between any two points of an electrical system, or between any point of such a system and the earth, as determined by a voltmeter.

"Potential" or "voltage" of a circuit or piece of electrical apparatus is the potential normally existing between the conductors of such circuit or the terminals of such apparatus. Where the difference in potential can not exceed 300 volts the supply shall be deemed a low voltage supply. Where the difference may at any time exceed 300 volts but can not exceed 650 volts, the supply shall be deemed a medium voltage supply. Where the difference may at any time exceed 650 volts, the supply shall be deemed a high voltage supply.

"Grounding" any part of an electric system shall consist in so connecting such part to the earth that there shall be no difference of potential between them.

"Explosion or flameproof casings or enclosures" are those which, when completely filled with a mixture of methane and air and the mixture exploded, are capable of either entirely confining such explosion within the casing or of so discharging the products of it that they can not ignite a mixture of methane and air combined in proportions most sensitive to ignition and entirely surrounding the points of discharge and in most intimate proximity therewith.

"Underground station" is any place where electrical machinery is permanently installed. (Id.)

341. Capacity, installation, etc., of apparatus.- Electrical apparatus and conductors shall be sufficient in size and power for the work they may be called upon to do, efficiently covered or safeguarded, and so installed, worked, and maintained as to reduce danger from accidental shock or fire to the minimum, and shall be of such construction and so worked that the rise in temperature caused by ordinary working will not injure the insulating materials. For underground work, when supplied with voltage higher than medium, no transformer shall have a normal capacity of less than 15 brake horsepower. (Id., sec. 1, rules 1, 2.)

342. Grounding.- All metallic coverings, armoring of cables, other than trailing cables, and where installed underground the frames and bedplates of generators, transformers, and motors, other than low-voltage portable motors, shall be efficiently grounded, as shall the neutral wire of 3-wire continuous current systems. (Id., rule 3.)

343. Voltage restrictions; danger signals.- Motors of coal cutting and other portable machines, and of electric locomotives, shall not be used at a voltage higher than medium. No higher voltage than medium shall be used underground except for transmission or for application to transformers or other apparatus in which the whole of the high-voltage circuit is stationary. In gaseous mines high-voltage transmission cables shall be installed in the intake airways only, and high-voltage motors and transformers shall be installed only in suitable chambers ventilated by the intake air which has not passed through or by a gaseous district. All high-voltage machines, apparatus, and lines shall be so marked as to indicate clearly that they are dangerous by the use of the word "Danger" placed at frequent intervals. (Id., rules 4-7.)

344. Ground detectors.- All underground systems of distribution that are completely insulated from earth shall be equipped with ground detectors. The condition of such system as indicated by the detector shall be noted each day by the person in charge of the underground wiring, or by another competent person who shall immediately report to him the occurrence of a ground. (Id., rule 8.)

345. Switchboards.- Main and distribution switch and fuse boards shall be made of incombustible insulating material free from metallic veins, and be fixed in as dry a situation as practicable. (Id., rule 9.)

346. Precaution against shock.- Gloves or mats of rubber or other suitable insulating material shall be provided and used by persons engaged in repairing the live parts of any electrical apparatus, or when the live parts of such apparatus have to be handled for the purpose of adjustment. (Id., rule 10.)

347. Electrician; damaging or interfering with electrical system.- At every mine where electricity is used below ground for power there shall be employed a competent mine electrician, who shall have full charge of the electrical apparatus in the mine, subject to the authority of the mine foreman. Any person wilfully damaging or without authority altering or making connections to any portion of the electrical system shall be guilty of a misdemeanor. (Id., rules 11, 12.)

348. Restoration from shock.- Instructions shall be posted in every generating, transforming, and motor room, and at the mine entrance, containing directions as to the restoration of persons suffering from electric shock, and all employees working in connection with electrical apparatus shall know how to carry out these instructions. (Id., rule 13.)

349. Plan of electrical system.- A plan shall be kept at the mine showing the location of all stationary electrical apparatus, including permanent cables, conductors, lights, switches, and trolley lines. The plan shall show clearly the position of such apparatus and the scale shall not be less than 200 feet per inch. There shall be stated on the plan the capacity in horsepower of each motor, and in kilowatts of each generator or transformer, and the nature of its duty. Such plans shall be corrected as often as necessary, at intervals not exceeding six months. (Id., rule 14.)

350. Report of defective equipment.- In the event of a breakdown, injury to electrical apparatus, overheating, or the appearance of sparks or arcs outside of enclosing casings, or in case any portion of the equipment, not a part of the electrical circuit, becomes alive, the same shall be promptly reported to the person in charge of electrical equipment. (Id., rule 15.)

351. Switchboards.- All switches, circuit breakers, rheostats, fuses, and instruments used in connection with underground motor generators, rotary converters, high-voltage motors, transformers, and low and medium voltage motors of more than 50-horsepower capacity, whall be installed upon a suitable switchboard. Similar equipment for low and medium voltage motors of 50 horsepower and less may be separately installed, if mounted upon insulating bases of slate or equivalent insulating material. In underground stations where switchboards are installed, there shall be a passageway not less than 3 feet wide in front of the switchboard, and if there are any high voltage connections at the back any passageway behind the board shall not be less than 3 feet clear. The space at the back of the switchboards shall be properly floored, accessible from each end, and in case of high voltage switchboards shall be kept locked up, but the lock shall allow of the door being opened from the inside without the use of a key. The floor at the back of high voltage boards shall be incombustible. Where the supply is at a voltage exceeding the limits of medium

voltage, there shall be no live metal work on the front of the main switchboard within 7 feet of the floor or platform, and the space provided back of the board shall not be less than 4 feet. Insulating floors or mats shall be provided for medium voltage boards, where live metal work is on the front. (Id., sec. 2, rules 16-19.)

352. Protection of underground circuits.- In every completely insulated feeder circuit in excess of 25 kilowatts capacity, leading underground and operating at a potential not exceeding the limits of medium voltage, there shall be provided above ground a switch on each pole and an automatic overload circuit breaker on at least one pole of direct-current circuits, and on at least two poles of polyphase alternating-current circuits. In case of ground-return direct-current circuits, a switch and circuit breaker shall be installed in the underground side of the circuit, but may be omitted from the return side. Fuses may be substituted for circuit breakers in circuits transmitting 25 kilowatts or less. Each circuit leading underground shall be provided with a suitable ammeter, Every alternating-current feeder circuit leading underground and operating at a potential exceeding medium voltage shall be provided above ground with an oil break switch on each pole, such switches to be equipped with an automatic overload trip. Each such circuit shall also be provided with a suitable ammeter. (Id., rules 20, 21.)

353. Transformers; transformer rooms.- Transformer rooms shall be of fireproof construction. Where the potential of circuits entering or leaving a transformer exceeds medium voltage, they shall be protected by an oil break switch, equipped with an automatic overload trip, on each pole. Where the potential does not exceed medium voltage, they shall be protected by a switch and an automatic circuit breaker on each pole, except that fuses may be substituted for circuit breakers in the case of lighting circuits and in the case of power circuits transmitting 25 kilowatts or less. All transformers shall be provided with suitable ammeter in either the primary or secondary circuits. (Id., rules 22-25.)

354. Protection of machine terminals; unauthorized persons; fire buckets.- All terminals on machines over medium voltage underground shall be protected with insulating covers or with metal covers connected to earth. No person other than one authorized by the mine foreman shall enter a station or transformer room, or interfere with the working of any apparatus connected therewith. Fire buckets, filled with clean, dry sand, shall be kept in electrical stations and transformer rooms, ready for immediate use. (Id., rules 26-28.)

355. Power and light circuits.- All high-pressure wires used inside of the mines shall be in the form of insulated, lead-covered, or armored conductors, subject to insulation tests, and with carrying capacity according to the rules of the National Board of Fire Underwriters. Medium or low pressure conductors may be bare, except that in gaseous portions of mines no bare conductors shall be used in rooms or beyond the last cut-through in intake entries. All underground cables and wires, other than trailing cables, unless provided with grounded metallic covering, shall be supported by means of efficient insulators. The conductor connecting lamp to the power supply shall in all cases be insulated. (Id., sec. 3, rules 29, 30.)

356. Main circuits.- Every main circuit coming from generating or transformer stations shall there be provided with switches, fuses, and circuit breakers, as described in paragraphs 352 and 353. If the transmission lines of low or medium voltage from the generating station are overhead, lightning arresters shall be installed in connection therewith at the generating station. If the distance from the generating station to the point where the lines enter the mine is more than 500 feet, an additional arrester shall be installed at this point, and in no case shall the arresters be more than 1,000 feet apart. In any gaseous mine, or gaseous portions of a mine, the electrical supply shall be brought underground only through such portions of the mine as are ventilated by intake air. (Id., rules 31-33.)

357. Size of conductors.- The size of all conductors shall be determined with regard to the maximum current which they are to carry, by reference to the table provided by the National Board of Fire Underwriters which shows the maximum current-carrying capacities of copper conductors. (Id., rule 35.)

358. Branch, grounded, and overhead circuits.- Every branch circuit shall be provided, at the point where it leaves the main circuit, with a switch of not less than 100-ampere capacity on each pole. One side of grounded circuits shall be efficiently insulated from earth. Overhead bare wires above ground shall be supported upon insulators which shall be adequate in quality, size, and design for voltage transmitted. (Id., rules 34, 36, 37.)

359. Underground trolley.- In underground roads the trolley wires shall be installed as far to one side of the passageway as practicable, securely supported upon hangers, efficiently insulated, and placed at such intervals that the sag between points of support shall not exceed 3 inches. The sag may exceed 3 inches if the height of the wire above the rail is 5 feet or more and does not touch the roof when the trolley passes under. All other wires, except telephone, shot-firing, and signal wires, shall be on the same side of the road as the trolley wire. At all landings and partings where men are regularly required to work or pass under trolley or other bare power wires, which are less than 6½ feet above top rail, a suitable protection shall be provided. All branch trolley lines shall be fitted with an automatic trolley switch or section insulator and line switch, or some other device, that will allow the current to be shut off from such branch headings. It is recommended that, where air or water pipes parallel the grounded return of power circuits, the return be securely bonded to such pipes at frequent intervals, to eliminate the possibility of a difference of potential between rails and pipes and to prevent electrolysis of the pipes. The rail return shall be of sufficient capacity for the current used, independent of the capacity of the pipes. On main haulage roads both rails shall be bonded, and cross bonds shall be placed at points not exceeding 200 feet apart. (Id., rules 38-42.)

360. Lighting circuits.- Where wires for electric incandescent lamps are connected to the trolley circuit, the lug of the trolley hanger to which connection is made shall be drilled to receive the lighting wire and provided with a set screw for securing same in place. Lighting wires shall not be wrapped or tied about the stems or studs of trolley hangers. The ground connection for lighting wires taken off the trolley circuit must be made to the track circuit. Wires for all lighting circuits shall be covered with an insulation adequate for the voltage of the circuit and strung on porcelain or glass insulators, unless they are encased in pipes or other metallic covering. If separate uncased wires are used they shall be kept at least 3 inches apart, except where they enter the fittings. If metallic casings are used they shall be grounded efficiently. (Id., rules 43-44.)

361. Joints in conductors; insulation and covering.- All joints in conductors shall be mechanically and electrically efficient, and where possible they shall be soldered. Wherever the conductors can not be soldered together, suitable screw clamps or connectors shall be used. All joints in insulated wire shall, after the joint is complete, be reinsulated to at least the same extent as the remainder of the wire. All high-voltage conductors inside of the mines shall be in the form of insulated, lead-covered, or armored cables, subject to approved insulation tests, and having carrying capacities in accordance with paragraph 357. Where lead-covered or armored cable is used, the lead or armor shall be electrically continuous throughout and shall be efficiently grounded. (Id., rules 45-47.)

362. Cables entering fittings; joints in cables.- The exposed ends of cables, where they enter fittings, shall be so protected and finished off that moisture can not enter the cable, or the insulating material, if of an oily or viscous nature, can not leak. Where unarmored cables or wires pass through metal frames, or into boxes or motor casings, the holes shall be substantially bushed with insulating bushings, and where necessary with gas-tight bushings which can not readily become displaced. Where cables other than signal cables are joined, suitable junction boxes shall be used, or the joints shall be soldered and the insulation, armoring, or lead covering replaced in at least as good condition as it was originally. (Id., rules 48-50.)

363. Power wires and cables in shafts.- All power wires and cables in hoisting shafts or manway compartments shall be highly insulated and substantially fixed in position. Shaft cables whose conductors or covering are not capable of sustaining their own weight shall be supported at intervals not to exceed 25 feet by suitable grips which can not cause abrasion of the covering or insulation. Where the cables are not completely boxed in and protected from falling material, space shall be left between them and the side of the shaft, so that they may yield and lessen a blow given by falling material. (Id., rule 51.)

364. Cables in haulage roads; protection during blasting.- Where the cables or feed wires, other than trolley wires, in main haulage roads can not be kept at least 12 inches from the mine car or locomotive they shall be specially protected by proper guards. Cables and wires, unless provided with metallic coverings, shall not be fixed to walls or timbers by means of uninsulated fastenings. When main or other roads are being repaired, or blasting is being carried on, suitable temporary protection from damage shall be given the cables. (Id., rules 52-54.)

365. Trailing cables.- Trailing cables for portable machines shall be specially flexible, heavily insulated, and protected with extra stout braiding, hose pipes, or other equally effective covering. Each cable in use shall be examined daily by the machine operator who shall also observe the trailing cable while in use and report any defect to the person in charge of electrical equipment. In the event that the trailing cable in service breaks down, becomes damaged, or inflicts a shock upon any person, it shall be at once put out of service, and shall not again be used until it has been repaired and tested by a properly authorized person. The cable shall be divided at the motor, but only for such length as necessary for making connection to the motor, and the cable with its outer covering complete shall be securely clamped to the motor frame in such a manner as to protect the cable from injury and to keep any mechanical strain off of the single ends that make electrical connection to the motor. In gaseous portions of mines a fixed terminal box shall be provided at the points where trailing cables are attached to the power supply, which box shall be flameproof and contain a switch and fuse on each pole of the circuit. The switch shall be so arranged that it can only be operated from without the box, when the latter is completely closed, and the switch shall be so constructed that the trailing cables can not be attached or removed when the switch is closed. (Id., rules 55-59.)

366. Switches, fuses, and circuit breakers.- Fuses and automatic circuit breakers shall be so constructed as effectually to interrupt the current on short circuit, or when the current through them exceeds a predetermined value. Open-type fuses shall be provided with terminals. Circuit breakers shall be adjustable to trip at from 50 to 150 per cent of their normal rated capacity, and provided with an indicator which shall show at what current the circuit breaker is set to trip. Fuses shall be stamped or marked or shall have a label attached, indicating the maximum current which they are intended to carry. Fuses shall only be adjusted or replaced by a competent person authorized by the mine foreman. Circuit breakers used to protect feeder circuits shall be set to trip when the current exceeds by more than 50 per cent the current-carrying capacity of the feeder. The circuit breaker may be equipped with a device which will prevent its acting unless the overload persists for a longer period than 10 seconds. Fuses used to protect feeders shall have a smaller current rating than the feeder. All switches, circuit breakers, and fuses shall have incombustible bases. All points at which a circuit, other than a signal circuit, has to be made or broken, shall be provided with proper switches.

The use of hooks or other makeshifts is prohibited, except that connection for gathering locomotives, or locomotives and machines used in driving headings or rooms, may be made to the trolley by suitable hooks. Switches shall be so installed that they can not be closed by gravity. In any gaseous portion of a mine, switches, circuit breakers, or fuses shall not be of the open type but must be inclosed in explosion-proof casings or break under oil. (Id., sec. 4, rules 60-65.)

367. Stationary motors.- Every stationary motor underground, together with its starting resistance, shall be protected by a fuse on each pole or circuit-breaking device on at least one pole for direct current and two for alternating current, and by switches arranged to cut off entirely the power from the motor. Such devices shall be installed near the motor. Every such motor of 100 brake horsepower or over shall be provided with a suitable meter to indicate the load on the machine. Motors used for operating fans in nongaseous mines where they are not under constant supervision shall be totally enclosed (but not necessarily explosionproof), unless placed in a chamber or passageway free from combustible material and completely lined with incombustible material. (Id., sec. 5, rules 66, 68.)

368. Motors in gaseous mines; detection of gas.- In any gaseous portion of a mine, all motors, unless placed in rooms separately ventilated with intake air, shall have all their current-carrying parts, starters, terminals, and connections completely enclosed in explosionproof enclosures made of noninflammable material. These enclosures shall not be opened except by an authorized person, and then only when the power is switched off. The power shall not be switched on while the enclosures are open. In working places where gas is likely to be encountered a safety lamp or other suitable apparatus for detecting fire damp shall be provided for use with each machine when working, and should any indication of fire damp appear, the person in charge shall immediately stop the machine, cut off the current at the nearest switch, and report the matter to the mine foreman. (Id., rules 67, 69.)

369. Inspection of enclosed motors.- All enclosed motors used underground shall be opened and inspected by the person in charge of electrical equipment, or his assistant, at least once a week, and where necessary shall then be cleaned and repaired. Enclosed switches shall be opened and inspected at least once a month. (Id., rule 70.)

370. Electric coal-cutting machines.- No man shall be placed in charge of a coal-cutting machine in any gaseous portion of a mine who is not a competent person, capable of determining the safety of the roof and sides and detecting the presence of explosive gas. In any gaseous portion a coal-cutting machine shall not be brought within the last break-through next the working face, until the machine man, or some other person appointed for that purpose by the mine foreman, shall have made an inspection for gas in the place where the machine is to work. If explosive gas is found the machine shall not be taken in. No coal-cutting machine shall operate in a gaseous portion longer than half an

by the mine foreman shall be allowed to fire shots electrically in any mine. All electric detonators and leads thereto shall be suitable for the conditions under which the blasting is carried on, and shall be of a type approved by the testing station of the Federal Bureau of Mines. Detonators shall be kept in a dry place and never stored with any other explosive. (Id., sec. 8, rules 87-90.)

374. Portable firing machines and batteries.- Portable shot-firing machines, sometimes called generators, shall be inclosed in a tightly constructed case when employed in any portion of the mine. All contacts, when made or broken, shall be within the case, except that the binding posts for making connections to the firing leads may be outside. Primary or secondary batteries used for shot firing shall be provided with a suitable case as above. The batteries shall be constructed so that, if the wires of the detonator or leads should accidentally come in contact with the binding posts no current will be discharged. They shall be provided with a detachable handle, plug, or key, without which the current can not be closed, or provided with one or more safety contact buttons, well counter-sunk or protected by a nonconducting housing. The plugs, handles, or keys shall be detached when not actually in use for firing a shot, and shall not pass from the personal custody of the person commissioned to fire the shots while on duty. All portable devices for generating or supplying electricity for shot firing when in a mine shall be in charge of the person commissioned to fire the shots. Such devices shall be frequently tested to insure that their capacity has not been decreased. No firing machine or battery shall be connected to the shot-firing leads until all other steps preparatory to the firing of a shot have been completed, and all persons have moved to a place of safety, and no person other than the shot firer shall make such connection. (Id., rules 91-94, 96.)

375. Disconnecting of firing leads.- Immediately after the firing of a shot the leads shall be disconnected from the source of electricity. No person shall approach a shot which has failed to explode until the firing leads have been so disconnected and an interval of five minutes has elapsed since the last attempt to fire the shot. (Id., rule 95.)

376. Use of special shot-firing systems.- The use of special electrical shot-firing systems or equipment not covered by the foregoing shall receive the approval of the testing station of the Federal Bureau of Mines. (Id., rule 97.)

377. Electric signalling; telephones.- Precautions shall be taken to prevent electric signal and telephone wires from coming into contact with other electric conductors, whether insulated or not. Bells, wires, insulators, contact makers, and other apparatus used in connection with electric signaling underground shall be of suitable design, of substantial and reliable construction, and erected in such a manner as to reduce the liability of failures or false signals to a minimum. In any gaseous portion of a mine, the potential used for signal purposes shall not exceed 24 volts, and bare wires shall not be used for signal circuits except in haulage roads. It is recommended that telephonic communication be established between the outside of the mine and the principal points of operation underground. (Id., sec. 9, rules 98-101.)

378. Electric relighting of safety lamps.- If in any place or portion of a mine in which safety lamps are used, they are relighted underground by electricity, the mine foreman shall select a suitable station, not in the return airway, where there is not likely to be any accumulation of inflammable gas; and no electric relighting apparatus shall be used in any other place. All electrical relighting apparatus shall be securely locked and shall not be available for use except by persons authorized by the mine foreman to relight safety lamps, and such persons shall examine all safety lamps brought for relighting before they are reissued. (Id., sec. 10, rule 102.)

Article 17. Accidents; Care of Injured Persons; Inquests

379. Injured persons to be given medical treatment.- If any person receives an injury in or about the mine and the injury shall come to the knowledge of the mine foreman, he shall see that the injured person receives treatment, if he is of the opinion that the injured person requires medical or surgical attention. In case of inability of such person to pay therefor, the expense shall be borne by the county. (1911, June 9, P.L. 756, Art. XXV, gen. rule 33.)

380. Ambulance and stretchers to be provided.- The operator or superintendent of every mine in which 50 or more persons are employed inside, shall provide and keep in good condition at the principal entrance of such mine, or at such other place as the superintendent and inspector may designate, one ambulance and at least two stretchers for conveying to his place of abode any person that may be injured while in the discharge of his duties, and also woolen and waterproof blankets. If the places of abode of 90 per cent of the workmen at any mine are within a radius of 1 mile from the principal entrance, an ambulance shall not be required, and where two or more mines are located within 1 mile of each other, or where the ambulance is located within 1 mile of each mine, only one ambulance shall be required if such mines have ready and quick means of communication by telegraph or telephone. The ambulance shall be constructed upon good substantial easy springs. It shall be covered and closed, shall have windows on the sides or ends, and shall be provided with spring mattresses or other comfortable bedding, to be placed on roller frames, together with sufficient covering and protection for the convenient movement of the injured. It shall be of sufficient size to convey at least two injured persons and two attendants at one time, and shall be provided with seats for the attendants. The stretchers shall be constructed of such material, and in such manner, as to insure ease and comfort in the carriage of injured persons. (Id., Art. XIII, secs. 1, 2.)

381. Medical supplies and suitable room to be provided.- At all mines there shall be provided bandages, splints, and other medical supplies, to render first aid and relief to injured employees. These supplies shall be kept in a suitable room, located near the entrance to or inside of the mine. The room shall be sufficiently large to accommodate the injured employees while they are receiving temporary medical attention. (Id., sec. 2.)

382. Removal of injured persons.- Whenever any person employed in or about a mine shall receive such an injury by accident or otherwise as to render him unable to walk to his place of abode, the operator or superintendent shall immediately have the person removed to his place of abode or to a hospital, as the case may require. (Id., sec. 3.)

383. Removal by conveyances other than ambulance.- If the injured person can be conveyed to his home or to the hospital more conveniently and more quickly by railroad, trolley road, or other conveyance, such mode of conveyance shall be permitted and no ambulance required; but in such cases the conveyance must be under cover and the comfort of the injured person must be provided for. (Id., sec. 4.)

384. Inspector and coroner to be notified of certain accidents.- Whenever a fatal accident occurs in or about a mine, or whenever an explosion or other serious accident of an unusual nature occurs, whether fatal or not, the mine foreman or superintendent shall give notice thereof forthwith by telephone or telegraph to the inspector, and also to the coroner of the county if any person is killed. (Id., Art. XXVII, sec. 1.)

385. Investigation by inspector.- Upon being notified of any fatal accident the inspector shall proceed as soon as practicable to the scene of the accident, and make such suggestions or give such directions as may appear necessary to secure the safety of any person who may still be endangered. Whether or not the results of the accident require an investigation by the coroner the said inspector shall proceed to investigate and ascertain the cause of the accident, and make a record thereof. To enable him to make the investigation he shall have power to compel the attendance of persons to testify, and also to administer oaths or affirmations. If it is found upon investigation that the accident is due to the violation of any of the provisions of this act by any person other than those who may be deceased, the inspector shall institute proceedings against such person. The costs of such investigation shall be paid by the county in which the accident occurred. (Id., secs. 3, 4.)

386. Inquests; inspector to be notified.- If the coroner shall determine to hold an inquest he shall notify the inspector of the time and place of holding the same, and the inspector shall offer such testimony as he may deem necessary thoroughly to inform the said inquest of the cause of the death. He shall have authority at any time to appear before such coroner or jury and examine or cross-examine any witness. No person who is directly or indirectly interested or employed by the person or company owning or operating such mine, or employed in or about any other mine in which such owners or operators may be interested, shall be competent to serve upon such coroner's jury. (Id., sec. 2.)

Article 18. Criminal Liability

387. Any person who shall intentionally or carelessly injure any safety lamp, instrument, air course, or brattice, or without proper authority handle, remove, or render useless any fencing, means of signaling, apparatus, instrument, or machinery, or shall obstruct or throw open airways, or enter a place in or about a mine against caution, or carry fire, open lights, matches, pipes, and other smokers' articles beyond any station inside of which locked safety lamps are used, or open a door in the mine and not close it immediately, or open any door the opening of which is forbidden, or disobey any order given in carrying out the provisions of this act, or do any other act whatsoever, whereby the lives or the health of the persons employed, or the security of the mine or machinery, are endangered, shall be deemed guilty of a misdemeanor, and shall be punished as provided in the following section. (Id., Art. XXVI, sec. 1.)

388. Penalty.- Any person who neglects or refuses to perform the duties required of him by this act, or who violates any of the provisions or requirements thereof, shall be deemed guilty of a misdemeanor, and shall upon conviction thereof be punished by a fine not exceeding $200, or imprisonment not exceeding three months, or both. (Id., sec. 2.)

CHAPTER IV - OIL AND GAS WELLS

Article 1. Care and Protection of Wells and Manner of Operation in General

389. Preventing pollution of springs, steams, etc.- Upon abandoning or ceasing to operate any well drilled for oil or gas, the person interested in such well shall plug it so as completely to shut off and prevent the escape of all water therefrom which may be impregnated with salt or other substances which will render such water unfit for use for domestic, steam making or manufacturing purposes, and in such manner as to prevent water from any such well from injuring or polluting any spring, water well, or stream which is or may be used for the purposes aforesaid. Violations of this act are a misdemeanor, punishable by a fine not exceeding $1,000, or imprisonment not exceeding six months, or both. (1891, May 26, P.L. 122, secs. 1, 2.)

390. Well may be plugged by persons injured.- In case any person is injured by failure to comply with the foregoing provision such person, after notice to the owner or lessee of the premises, may enter upon and plug the well, and thereupon may recover the expense thereof from the person by whom it should have been plugged. (Id., sec. 3.)

391. Manner of plugging abandoned wells; vent pipes in abandoned wells bored through coal.- Upon abandoning or ceasing to operate any well drilled on lands within this Commonwealth for exploring for or producing oil or gas, the person or corporation drilling or owning the wells shall plug said wells in the following manner: Fill up the well with rock sediment to a point 20 feet above the top of the lowest oil or gas bearing strata or formation encountered, and drive a round, seasoned, wooden plug, at least 3 feet long equal in diameter to that of the well below the casing, and shall in like manner keep plugging and filling until all producing sands have been plugged, when a final plug must be anchored approximately 10 feet below the bottom of the largest casing and filled in with such an amount of rock or rock sediment as may be necessary to completely shut off any water bearing sands or strata; the fill, however, shall in no event be less than 30 feet deep. All plugs used shall be well-seasoned, round, wooden plugs of the diameter of the well at the point at which the plug shall be located. They shall be at least 3 feet long and shall have the lower end tapered for a distance of 18 inches. In abandoning any well drilled through marketable coal, after the inside casing has been drawn, there shall be placed a 2-inch vent pipe extending from a point 30 feet below the coal bed for a distance of 80 feet in height; the pipe must be encased in cement, after which the outside casing may be drawn. (1921, May 17, P.L. 912, sec. 1.)

392. Pulling of casings in gas wells; plugs or bridges upon abandonment of such wells.- Whenever the production of any gas well is not sufficient to justify leaving the casing in the well, the well may be utilized through tubing placed on a packer, and after cement and sand have been poured on the packer to a depth of not less than 10 feet, the casing may be pulled and the hole outside

of the tubing filled with sand, cement, rock, sediment, clay, or other material to a point at least 30 feet above the highest producing sand, so as completely to shut off all water from the various sands, after which the casing may be drawn. Upon abandonment of such gas well, if a plug or bridge shall be placed in the tubing at a point near the depth at which the packer was set, and cement and sand poured in on said plug or bridge to a depth of not less than 30 feet, it shall be held a compliance with the provisions of this act relating to plugging and abandoning of wells. (Id., sec. 2.)

393. Uncapped wells; opening valves; opening wells to clean, etc.- The wilful permitting of any oil or gas well to remain uncapped, or the wilful opening of any valve to admit air in a gas-pumped territory in which the gas pressure is less than atmospheric pressure, is declared to be a misdemeanor. In case any well in such territory is opened for cleaning, repairing, abandoning, etc., the same must not remain open continuously longer than 12 hours, unless work in connection with cleaning, etc., is being conducted more than 12 consecutive hours. (Id., sec. 5.)

394. Oil or gas wells producing from certain strata; keeping open to permit operation by pressure.- The owner or operator of any well or wells which produce oil or gas from the strata known as the Bradford sand, Ball Town sand, Clarendon sand, Cherry Grove sand, Kane sand, Glade sand, and Haskell sand, shall be permitted to allow such wells to remain open for the purpose of introducing air, gas, water, or other liquid pressure upon the sands for recovering the oil and gas contained therein, provided that the introduction of such pressure into the sands shall be through casing or tubing which shall be so anchored and packed that no other oil or gas bearing sand, above or below these particular sands, shall be affected by the introduction of such pressure. (1929, Apr. 26, P.L. 821, sec. 1.)

395. Rights of adjoining owners with respect to plugging or capping abandoned well.- Whenever any owner or operator shall neglect or refuse to comply with the provisions of this act, the owner of or operator upon any adjoining or contiguous land may enter, take possession of said abandoned well, and plug or cap it as provided by this act, and recover the expense thereof from the owner or operator whose duty it may have been to comply with the provisions of this act. (1921, May 17, P.L. 912, sec. 7.)

396. Drilling on lands near productive well; duty of driller.- The owner or operator of any well productive of oil or gas in paying quantities shall have the right to give written notice to any other owner or operator who may be about to drill or may be drilling a well within 1 mile of the productive well, that the well about to be drilled or being drilled will penetrate the same sand or strata as that from which the productive well obtains its production. After the service of such notice the owner of the well being drilled or about to be drilled shall case off all water found therein if the water can not be bailed out of the hole, while drilling, with the use of an extension bailer, so as to prevent the entry of the water into the sand or strata from

which the oil or gas is obtained in the productive oil or gas well. No such notice shall be of any effect unless given before the sand to be protected shall have been penetrated by the well about to be drilled or at the time it is being drilled. (Id., sec. 4.)

397. Violations of act; penalty.- Any person or corporation violating the provisions of this act shall upon conviction thereof be sentenced to pay a fine not exceeding $1,000 or undergo imprisonment not exceeding one year. (Id., sec. 6.)

398. Penalty for willfully injuring or destroying oil or gas wells, etc.- Any person willfully and maliciously injuring or destroying, or attempting to injure or destroy, any oil or gas well, or any reservoir, or tank, used for the storage of oil or gas, or any pumping station, valve, or pipe line used for the transportation of oil or gas, or any machinery connected with such wells, tanks, pipe lines, etc., shall be guilty of a misdemeanor, punishable by a fine not exceeding $1,000 or imprisonment not exceeding three years, or both. (1917, July 6, P.L. 748, sec. 1.)

Article 2. Protection from Fire

399. Burning of brush, etc.- Fallows, stumps, brush, etc., shall not be burned in any forest lands of this Commonwealth in which there are producing oil or gas wells, or rigs erected for drilling such wells, from April 1st in each year to May 20th next ensuing, nor from September 10th to November 10th next ensuing. Excepting during the periods aforesaid, fires may be set in such lands upon compliance with conditions prescribed in the act. (1907, June 12, P.L. 527, sec. 1.)

400. Removal of brush, tree tops, etc.- Any owner or lessee of any forest lands, or owner of trees growing upon such lands, or any person in charge of the premises upon which lands there are producing oil or gas wells, or rigs erected for drilling such wells, shall at least once each year cause to be moved from said lands all brush, tree tops, and branches of trees, which such person may have cut or felled thereon, within 100 feet of all such wells or rigs; and shall at least once in the year cause to be removed from said land all grass, brush, and other inflammable material, within 100 feet of the right of way of any railroad company operating thereon; to the end that during the periods defined in the above paragraph, the said area shall be clear of such inflammable material. (Id., sec. 2.)

401. Removal of inflammable material from railroad right of ways; precautions to be taken by railroads.- Every railroad company shall, on such part of its road as passes through forest land on which there are producing oil or gas wells, or rigs erected for drilling such wells, cut and remove from its right of way through said lands, at least once a year, all grass, brush and other inflammable materials; employing, in the seasons defined in paragraph 395, enough trackmen to put out promptly any fires on its right of way; provide locomotives thereon with steel netting or iron wire on the smokestacks, or other efficient spark arresters, and adequate devices to prevent the escape of fire from ash pans and furnaces, and the same shall be used by every engineer and fireman on such part of its road. No railroad company or employee thereof shall deposit fire, coals, or ashes on its track or right of way near such lands. In case of fire on its own or neighboring lands, within 100 feet of its tracks, the railroad company shall use all practicable means to put the fire out. (Id., sec. 3.

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Scott Turner, Director

INFORMATION CIRCULAR

MINING LAWS OF NORWAY

BY

E. P. Youngman

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE--BUREAU OF MINES

MINING LAWS OF NORWAY[1]

By E. P. Youngman[2]

PREFATORY NOTE

This paper is one of a series of digests of foreign mining legislation and court decisions that is being prepared in advance of a general report relative to the right of American citizens to explore for minerals and to own and operate mines in various foreign countries. In the preparation of this interpretation of the laws of Norway, use has been made of William R. Orrmins' translations of various acts of the Norwegian Government, which were forwarded from the American consulate at Oslo[3] and transmitted to the Bureau of Mines through the courtesy of the State Department. This digest is released subject to amplification and correction, if necessary, by the proper American foreign-service officers.

INTRODUCTION

Translations of the following Norwegian enactments were available:

1. Law concerning mine exploitation, of July 14, 1842 (including the modifications and amendments of September 24, 1851; September 28, 1857; May 22, 1902; May 9, 1903; March 11, 1905; June 13, 1924; and May 22, 1925).

2. Law dealing with the right to gather gold, etc., of June 17, 1869.

3. Law concerning the acquisition of waterfalls, mines, and other immovable property, of December 14, 1917.

4. Law governing ore out-croppings encountered within a section of Northern Trondhjem district, of May 31, 1918.

5. Law concerning the application of an orderly legal procedure, No. 4 of August 14, 1918.

6. Law dealing with amendments to legislation regulating concessions, of July 17, 1925.

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1 - The Bureau of Mines will welcome the reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6654."
2 - Rare Metals and nonmetals division.
3 - Philip, Hoffman, ________: Despatch 52, Oslo, Feb. 25, 1931.

7. Law concerning changes made in the law of December 14, 1917, on the acquisition of waterfalls, mines, and other immovable property, of June 25, 1926.

Laws 1 and 3 of the foregoing enumeration are the basic mining laws, the former dealing with prospecting and mining under permits and claim-staking patents and the latter with mining under concessions. All references in this paper to these two laws are designated by the year alone--that is, the law of 1842 or the law of 1917; references to the other laws carry the full date.

GENERAL

Prospecting, by either natives or foreigners, may be done on either State land or private land; the prospector may obtain written permission from the landowner to prospect, or he may obtain a permit from the proper official and present it to the landowner (or the user) of private land or to the public official in charge of State land. (Par. 1 and 2, law of 1842.) A national (individual or group) may obtain mining rights through a concession or through a claim-staking patent (letter of allotment), received after registration and claim staking. A foreigner, in order to carry on further operations, must acquire a concession. (Par. 11 and 12, law of 1917.)

The Government's authority over mining is exercised, in general, through the Department of Commerce. The officials charged with the issuance of prospecting permits are the local magistrates of district or town. (Par. 2, law of 1842.) The Superintendent of Mines issues claim-staking patents. (Par. 9 and 10, law of 1842.) Stipulations in each concession are laid down by the King. (Par. 13, law of 1917.)

The Superintendent of Mines shall not reject applications for claim-staking rights (or for surveys or respites) unless it is definitely ascertained that the applicants are not entitled thereto or unless strong doubt exists as to their eligibility. (Par. 16, law of 1917.)

The concession law provides that the Department of Commerce "may for the purpose of its control over forthcoming developments appoint a comptroller, who in accordance with instructions furnished him by the Department shall have full access to all means necessary for the exercise of such control. If any question arises as to the authority and duties of the comptroller, the concessionaire may demand a decisive statement from the Department. Concerning duties that do not have a bearing on the matter, he may guard silence as to the information reaching him." His compensation shall be established by the Department and be reimbursed into the public funds by the concessionaire. If a general comptroller is appointed for several mining establishments for certain parts of the country, the individual establishment may be assessed a proportional contribution toward the expenditures that the control by such an official entails. (Sec. 12, par. 13, law of 1917.)

RIGHTS OF FOREIGNERS

Every foreigner, in order to mine, must apply for and obtain a concession, whereas the Norwegian State, a commune, a citizen, a Norwegian corporation, institution, commercial company, or other company of limited liability, having a Norwegian board of directors with residence in Norway, may exercise, <u>without a concession</u>, the privileges inherent in the law governing mines. The law concerning the acquisition of waterfalls, mines, and other immovable property, December 14, 1917 (sometimes termed "the concession law"), reads as follows:

> Nobody except the State, the Norwegian community units, and Norwegian national citizens, as well as corporations, institutions, commercial companies, and other associations with a limited liability entirely under Norwegian management and with headquarters in Norway, shall exercise without a concession the privileges enumerated in the law governing mines for the purpose of organizing, registering, and claim staking . . . or in some other manner acquiring with full legal effectiveness the property right or exploitation right to licensable allotments or mine pits . . . subject to mining legislation, whether on owned lands or on those belonging to others. (Par. 11.)

> Without a concession, nobody except the State and the Norwegian community units shall commence the regular operation of mines on claim-stakeable ore or metal properties within the country. (Par. 12.)

> Norwegian and <u>foreign national citizens</u>, as well as corporations and other commercial companies and associations with a limited liability, societies, and institutions may, provided general interests do not prohibit, acquire concessions for the selection and operation of claim-stakeable (licensable) allotments or mine pits, on the basis of more detailed conditions laid down by the King. (Sec. 1, par. 13.)

Section 1, paragraph 13 of "the concession law" provides further, as follows:

> The concession is conferred upon a definite person, company, corporation, or institution. As a rule, it should be insisted upon that the management of company, corporation, or institution have its seat in Norway and that it be composed, at least in part if not entirely, of Norwegian national citizens. In a concession granted to a foreigner, it may be stipulated that Norwegian capital shall be admitted. In the case of a company, conditions may be included in the concession to prevent a majority of those making up the company or holding the shares thereof from making the concession the property of foreign citizens, foreign companies, or some other party that is already carrying on mining in the country or that owns a majority of the shares of some other company engaged in mining in Norway.

Furthermore, it should be a rule that, when regular operation has begun, a sum equal to at least one-third of the expenditures required for the acquisition of the mining property and for the necessary installations shall be available as working capital. In addition, the company may be required to offer as security an amount considered sufficient to assure activity for at least three months. This security shall not be touched except by express consent of the Department of Commerce. It shall, if infringed upon with such consent, be replaced, in accordance with specific provisions made by the Department of Commerce.

It will be seen from the foregoing paragraphs that the law does not specifically discriminate against foreigners but that it affords Government authorities the right to interpret the spirit of the law at their discretion in each instance. After a concession is granted, the Government is often criticized by the ultranationalistic press for permitting foreigners to gain control of Norwegian industries. On the other hand, intelligent business men appear to favor bringing in foreign capital;[4] and with respect to proposed changes in the concession laws, an editorial in a leading economic publication of Oslo expresses the following sentiment:[5]

> Although it is very important that we keep our economic independence, it does not seem advisable to build steel walls around us, if there is danger that they may finally be nothing more than an enclosure for a poorhouse.

The present concession laws are based on a definite policy of preventing foreigners from acquiring excessive control of natural resources, and contemplated changes (proposed by the cabinet and approved by the King, although not as yet by the parliament) very definitely confirm this policy.[6]

Section 38 of the proposed law further protects the Norwegian Government in cases where shares in Norwegian properties are held by Norwegian citizens but in reality are owned by foreigners. Such transactions may be declared invalid, if "it appears probable" that an attempt has been made to disguise ownership.

With respect to the employment of foreign labor in mining operations under a concession, section 4 of paragraph 13 of the concession law says:

> In the course of the layout and exploitation work of a mining property only those laborers and officials (clerks, foremen, superintendents, managers, etc.) shall as far as possible be employed who are natives of Norway or in possession of Norwegian national citizenship rights (par. 92-a, b, and d, Constitution). However, provided public

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4 - Ibid.

5 - Lund, Marquard H., Proposed Changes in Norwegian Concession Laws: Econ. and Trade Notes No. 205, Oslo, May 1, 1931, div. com. laws, Bur. For. and Dom. Com. file 129375.

6 - Ibid.

interest is not contrary thereto, foreign laborers may likewise be permitted to work, if they have established a fixed residence in Norway during the previous year. At the same time, Norwegian materials, Norwegian workmanship, and Norwegian insurance facilities are to be given the preference. More detailed provisions regarding the application of these conditions shall be incorporated in the concession document.

Section 10 of paragraph 13 of the concession law states that the concession may specify that the "treating process of the extracted mining products" shall be as far as possible carried out within the confines of Norway, and that Norwegian industrialists shall have access to the purchase of a certain percentage of the output at the same prices and under the same conditions as would be attained by a free sale elsewhere. The Department of Commerce will in certain instances make special regulations with regard to the manner in which these provisions shall be carried out.

A report[7] from an American representative in Norway in 1929 stated that there had been a marked increase in the activities of Norwegian mining companies during the preceding few months, owing to a certain extent to the investment of foreign capital. Among the recent acquisitions of Norwegian mining property by foreign capital was the sale of A/S Heraker Elektriske Kraft & Smelteverk to the Dominion Metallurgical Co. of Canada and the sale of Nikkel Raffineringaverket (in Kristiansand) also to Canadian interests. The Norwegian company, however, was retaining its mines and smelters, only the refinery changing hands. Three French experts were engaged in inspecting the properties of "A. B. Sulitelma gruber," and it was rumored that a sale to foreign interests was imminent. Both English and American capital was manifesting a desire to buy, but the management admitted nothing definite.

A number of years ago the National Lead Co. of America obtained control of the Titan Co. A/S (Fredrikstad, Norway) and of its ilmenite mines at Raegefjord.

INCORPORATION

Concerning the incorporation of foreign companies in Norway, the division of commercial laws, of the Bureau of Foreign and Domestic Commerce, published the following summary:[8]

> Under the Norwegian law, a foreign company, properly organized in its own country, may open a branch office and engage in legitimate business under the management of a special board of directors, subject to the provisions of the Norwegian law. The board must file a notification, duly executed, in the office of the Commercial Registrar, "Handelsregister," that the capital is entirely paid up.

7 - Lund, Marquard H., __________: Econ. Trade Notes No. 37, Oslo, Apr. 25, 1929. European sec., div. reg. infor., Bur. For. and Dom. Com., file 70000-Norway.
8 - Lund, Marquard H., Taxation and Company Law in Norway: Spec. Circ. No. 251, div. com. laws, Bur. For. and Dom. Com., Dec. 11, 1930, p. 2.

The notification must be signed by the members of the board of directors and must be accompanied by a certified statement from proper foreign authorities and from the Norwegian consul, to the effect that the company is duly incorporated in its own country and has its head office there. The members of the board are individually and collectively liable if the notification is neglected. In case of any change in the organization, bankruptcy included, the registrar's office must be notified. Only the amount paid up can be designated as the capital of the company.

The name of the foreign company must always be followed by the words "foreign share company" (Utenlands Aktieselskap) on stationery, advertising matter, and all documents.

Foreign corporations, issuing shares "to bearer," must publish the company's annual balance sheet at the end of each year in the official Norwegian publications, Norsk Kudgjerelsostidende.

Although foreigners may own the majority of the voting stock, the board of directors of a corporation, incorporated under Norwegian laws, must be Norwegian citizens or resident foreigners, who may be elected to membership on the board only after a residence of two years.

After the company is properly organized, the managing director must obtain a license (handelsbrev), or if the manager is not on the board of directors, at least one member of the board must obtain such a license. The person seeking a license must be a resident of Norway, must continue to reside there, must be 21 years of age, and must have a certain proficiency in bookkeeping and accountancy.

Legal assistance should be obtained, of course, in every case.

OWNERSHIP

The State has reserved to itself the right to mine minerals in the territory of the Northern Trondhjem district "that is bounded on the north by the district boundary of Nordland district, on the east by the National boundary, and on the south by a straight line running from the boundary islands No. 192 to the eastern inlet of Sandsjøen, to Kalviken, thence to the Sandsjøen, Laksjøen, Skjelbredvand, and Ottersjøen Lakes, with intervening land fractions, and thence to Sanddøla and Storelven, up to a point where it is traversed by the district boundary." (Par. 1, law of May 31, 1918.) The King may, with the consent of the Norwegian Parliament, grant contracts for mining in this locality to Norwegian communities, Norwegian national citizens, or corporations or other companies of limited liability with headquarters in Norway, with an all-Norwegian directorate and an all-Norwegian capitalization. (Par. 2, law of May 31, 1918.)

In the territory of the State throughout the Finnmark district, mining operations may be carried on only under provisions laid down by the King.

(Par. 2, law of June 17, 1869.) The King may, in the event of a demand for land or water rights belonging to the State in the Finnmark district or in some other lands of the State (where the original property rights are vested in the State) permit the claim staker to apply for an exploitation permit from the Arbitration Commission or the proper court, in accordance with rulings affecting the general rights of national citizens, enacted July 9, 1851. (Par. 3, law of March 11, 1905.)

Paragraph 75 of the law of 1842 exempts from the provisions of the law the Kongsberg Sølvverk.

> As long as this silver mine remains the property of the State, it shall retain for itself the exclusive right to operate silver and gold in the Sandsvaer, Flesberg, and Eger parochial districts, to the extent that no private party may obtain claim-staking patents within the districts mentioned.

The right to wash out gold or to gather in some other manner alluvial gold shall be vested in the landowner alone. The King shall determine how gold may be won on State land. (Par. 1, law of June 17, 1869.)

With the exceptions noted in the preceding paragraphs, "all metals and ores, save bog iron (bog ore) and sea ore, are open to prospecting." (Par. 8, law of 1842.) The minerals do not belong to the owner of the land, although he is entitled to participate with the discoverer of minerals in the exploitation thereof to the extent of one-tenth (the rule to apply to reopened mines, to abandoned mines, and to leftovers in abandoned mines as well). (Par. 14, law of 1842.)

PROSPECTING

Prospecting Permits

General permit.-- An ordinary prospecting permit is issued free of charge by the federal officials of the respective district or for 16 shillings by the "sheriff or rural mayor" of the respective district or by the local magistrate for work on town or city ground. A permit may be granted only to an individual personally known to the issuing authority, attested to by the priest, or proved in some other way to be responsible. (Par. 2, law of 1842.) Any one seeking to prevent legalized prospecting shall be liable to a fine of from 4 to 20 specie dollars and shall reimburse the prospector for any expenditures he may cause him to make. (Par. 6, law of 1842.)

Exclusive permit.-- If a discoverer desires to take exclusive advantage of his find, he shall notify the proper official in writing (in duplicate). (Par. 7, law of 1842) The petition shall specifically locate the discovery and shall name two witnesses thereof. The "rural mayor" or the magistrate will insert on the petition the year, date, and time of its presentation, together with a notation of any similar petition offered for the same location during the preceding 12 months. The petitioner shall pay the stamp fee

assessable under the law (law of May 22, 1925, No. 2), in behalf of the State treasury. One of the two forms filed, containing a stamp-fee receipt, shall be returned within at least 14 days to the petitioner. The other form shall be forwarded to the Superintendent of Mines, with a notation that the fee has been paid. The petitioner shall then, by public notification or in the presence of two witnesses, warn the property owner of the acceptance of the petition. (Par. 7, law of 1842.)

Duration.--Seemingly, a permit is granted for an indefinite period. (Par. 2, law of 1842.)

Exempted areas.-- No prospecting shall be done in cultivated lands ("aker og eng"), or within 100 ells from dwelling houses and outhouses, or in general anywhere except in open fields, without the express permission of the owner or user. No prospecting shall be done on the public highways. (Par. 3, law of 1842.)

Safety measures.-- If prospecting is carried on at a distance of less than 200 ells from a public highway or from a dwelling belonging to another person, the prospector shall, under penalty of a fine of 10 specie dollars,[9] post watchmen on the highway whenever blasting is done--one watchman on the road at each end of the prospecting area (at the distance previously mentioned). (Par. 3, law of 1842.)

The prospector, as soon as his excavations imperil domestic animals, must provide adequate fencing. If he abandons workings, he must fill in the hole or surround it with a solid stone bulwark or be liable to a fine of 10 specie dollars. (Par. 4, law of 1842.)

Damages.-- A prospector shall post security for any damages he may cause in prospecting, in the sum prescribed by the official issuing the permit (par. 2, law of 1842); and he shall make good any such damage (par. 3, 4, and 5, law of 1842).

MINING

Claim Staking

Claim-staking patents for new discoveries.-- The right to a claim-staking patent lies with the first discoverer, the land allotment being made available to others only in case the first finder fails to present an application.

An application for a patent must be written, must be accompanied by a sample of the mineral located, and must be presented to the Superintendent of Mines not later than eight months after the report of the discovery of the deposit has been made.

9 - Norwegian spesiedaler.

The Superintendent of Mines shall at once insert on the application for a patent the day and the hour it was received and shall thereupon make out a patent, containing the name of the applicant, a description of the mineral samples submitted, and the name of the landowner. While the application is pending, no prospecting may be done on another person's land that is in such close proximity to the recorded location that a possibility might arise of the owner's laying claim to the same mineral find. (See also section entitled "Survey of Mines.") (Par. 9, law of 1842.)

Claim-staking patents for old mines.-- Any one desiring to attempt the operation of abandoned mines or prospects shall make application in the same manner and under the same rules as though he were applying for a new location, except that the period allowed between a declaration of intention and the filing of an application shall be 18 months. (Par. 10, law of 1842.)

Publication and recording.-- At the end of each 3-month period, the Superintendent of Mines shall publish in the Norsk Kundgjørelsestidende a list of all claim-staking patents issued during the three preceding months. (Par. 11, law of 1842.)

When the Superintendent of Mines has approved a claim staking, he shall deliver to the assistant superintendent a confirmation and a copy of the patent. (Par. 13, law of 1842.)

Survey.-- Any one obtaining the right, through claim staking, to exploit an allotment may demand an official survey (or staking off), which shall be made within a year, and which shall give an exclusive privilege to mine within the staked area. (Par. 21, law of 1842.) Further details are covered by paragraphs 23 to 27, law of 1842.

Area.-- The law does not definitely limit the size of the area granted by a claim-staking patent (mining license). The section upon official surveys (chap. 4 of law of 1842) reads as follows:

> For metals and ores that are isolatedly gathered on sites of mineral deposits, the applicant may be granted, in accordance with his own request, as much as 150 fathoms (Norwegian) lengthwise, depending on the lay of the land, and 3½ fathoms on each side thereof, measured at right angles from the center of the mineral deposit. . . . .
>
> In so far as the area of the deposit is not taken up to its full length, because of the lay of the land, it shall be marked off as far as possible, and the right to pursue that area to its full length from the point of its departure shall remain with him to whom the allotment is granted.
>
> Whenever several areas adjoin each other so closely that they may be worked under one single management, all these areas shall be considered as only one allotment when surveyed.

With respect to metals and ores the outcroppings of which appear in other areas, the survey shall take the shape of a square, measuring not more than 2,500 fathoms. The "confines of such an allotment shall be carefully measured and shall apply perpendicularly." (Par. 23, law of 1842.)

The point where the mineral deposit is discovered shall lie within the allotment; otherwise the allotment may be laid out in any direction desired by the applicant, as long as the rights of other mine owners are not infringed upon (circumscribed by paragraphs 3 and 18).

Allotments laid below the surface shall be governed by the rules set forth in the foregoing paragraphs, after the applicant has named a point of departure. (Par. 24, law of 1842.)

Easements, et cetera.-- The holder of a claim-staking right is entitled to request the landowner to grant adequate room for roads and footpaths leading to the mine, tool and supply storehouse, and surrounding buildings "during the day" and to the water supply. The patent holder shall likewise be entitled to a sufficient supply of water, if it is available and not already required for some other industrial purpose from which it can not be withdrawn without injury to that industry. However, in those places where prospecting may not be undertaken without the consent of the landowner, no requirement shall be made except for the land itself and whatever roads may be necessary to give access thereto, and no demand shall be made for water from a source on adjoining property.[10] (Par. 18, law of 1842.)

Whatever a landowner may cede, in accordance with the foregoing provisions, shall revert to his property when no longer needed for the purpose for which it was ceded.[11] (Par. 19, law of 1842.) The landowner shall be fully compensated for the cession. Any disagreement with respect to the terms shall be decided by legal arbitration. This rule applies not only to the amount of the pecuniary compensation but also to the necessity for and the extent of the cession. The landowner may choose between a lump sum and yearly installments. If he is not occupying the land, the actual user thereof shall receive compensation in proportion to the time he holds the land; or if the landowner chooses to take compensation in a lump sum, the user shall be entitled to the dividends therefrom.[12] (Par. 20, law of 1842.)

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10 - The law of May 22, 1902, No. 8, reads: "The King or whoever presents an authorization may, upon request, permit the provisions of paragraphs 18, 19, and 20 of the law of July 14, 1842, to be fully applied, if it should prove necessary to construct general roads or railways for the transportation of the ore from the mine pit to the concentration shop or place of export.

11 - See footnote 10.

12 - See footnote 11. (Consult also the law on the acquisition of waterfalls, mines, and other immovable property of Sept. 18, 1909, par. 31, sec. 3.)

Whenever it proves advisable to construct auxiliary excavations (artificial or ventilation shafts) that will necessarily go beyond the surveyed allotment, the Superintendent of Mines shall, at his discretion, grant permission for such excavations if they can be made without creating openings in exempted areas. (Par. 40, law of 1842.) The landowner shall receive compensation, according to the rules set forth in the preceding paragraph. When operations are carried into the land of another person, he shall likewise be entitled to compensation. (Par. 40, law of 1842.) If the auxiliary excavations are carried into the mineral deposits of some one else, he shall be entitled to one-half of the benefits that may accrue therefrom. "With respect to the discovery of metals or ores below the surface of a claim-staked area, the provisions of paragraph 10 (concerning claim-staking patents for old or abandoned mines) shall apply." (Par. 41, law of 1842.)

Participation rights of landowner.-- The owner of land that is subject to a claim-staking patent is entitled to participate in the exploitation of the deposit to the extent of one-tenth. (This rule applies to old workings also.) (Par. 14, law of 1842.) As soon as the owner has expressed his desire to take advantage of his right of participation or to transfer that right to another, he is entitled to demand that one-tenth of the material already won (or of the profit therefrom) shall remain available until such time as the proper arrangements can be made. (Par. 15, law of 1842.)

If the landowner does not follow up his interests, as indicated in the following regulations, his share goes to the claim staker. (Par. 16, law of 1842.)

The landowner shall declare, before the expiration of six months from the date on which the patent was made known to him, his intention to participate and shall draw up a contract governing the conditions mutually agreed upon. If he does not desire to participate, if he can not come to an agreement with the claim staker, or if he is opposed to the terms offered by the Superintendent of Mines, he shall be allowed a period of three months more in which to reclaim his share at public auction. The auction agreement shall contain a provision that if the owner does not come to an agreement with the claim staker within one month as to the conditions that shall govern the operation of the mine, the owner shall be forced to accept terms laid down by the Superintendent of Mines, who shall be entitled to remuneration (according to a law of March 11, 1905, with respect to payments to public officials.) (Par. 16, law of 1842.)

The landowner (or his transferee) shall participate in all exploitation expenditures likewise. (Par. 17, law of 1842.)

Suspension.-- Unless a respite has been granted (upon application made therefor one month in advance), a claim-staked mine pit or excavation not kept continuously active shall become available to others.

A respite may not be granted during the first year after a claim staking has been authorized, nor in the case of unworked sections during the winter months of the first three years. (Par. 30, law of 1842.)

The Superintendent of Mines grants a respite (suspension) only when he has become satisfied that a request has been made for sufficient reasons beyond the control of the mine owner, owing to temporary conditions or local peculiarities connected with the operation of the mine or to temporary obstructions in connection with ore treatment or ore transportation. Likewise, a respite may be granted if it is proved that the mine pit or excavation must be set aside as a reserve or that even when inactive it is a part of the rest of the mine. (Par. 31, law of 1842.)

A respite may be renewed as often as legal reasons therefor exist. (Par. 32, law of 1842.)

The decision of the Superintendent of Mines (either refusing or granting a respite) may be appealed by the mine owner or by an interested third party and submitted to royal review. If a respite is granted by royal command, the date of that command shall apply. (Par. 33, law of 1842.)

Further details are covered by paragraphs 34 to 39, inclusive, of the law of 1842.

Abandonment.-- The former owner of an abandoned mine shall be entitled to make use of the tailings or dumps for a period of one year after abandonment, even if some one else recommences mining operations. (Par. 48, law of 1842.)

If a mine owner desires to abandon an active mine that is more than 10 fathoms deep, he shall signify (in writing) his intention so to do two months in advance to the Superintendent of Mines and shall deposit with him a chart covering the mining property. A mine owner defaulting in this provision shall be liable to a fine of 10 specie dollars. (Par. 49, law of 1842.)

An operator that has either abandoned or forfeited his mine may take away machinery or movable buildings if he is not able to reach an agreement concerning them with the succeeding operator; he may in no way damage the property or remove any of the construction work in or at the mine. (Par. 50, law of 1842.)

Concessions

As has been stated previously in this paper, a national may and a foreigner must operate under a concession. (Par. 11, 12, and 13, law of 1917.) (See section entitled "Rights of Foreigners.")

A concession affords the right to exploit mine pits or allotments, in full accordance with the concession terms, any legislation in force concerning mine exploitation, and especially the law of July 14, 1842. (Sec. 2, par. 13, law of 1917.)

Section 3 of paragraph 13 of the concession law of 1917 seems to carry prospecting (or at least preliminary mining) provisions. It reads as follows:

Preparatory work in anticipation of the commencement of exploitation shall begin within a fixed time limit. The work shall be done by technical methods, in accord with legislation laid down therefor. Should the Department of Commerce decide that the work deviates from customary mining procedure, and should representations to the mine management prove ineffective, the department may demand that the mine owner submit a schedule covering a certain period of time, and showing a definite plan for future operations. This schedule shall be presented to the department not later than four months after the official request therefor has been made. Further exploitation thereafter shall take place only in conformity with the schedule approved by the department. The question as to whether exploitation is in conformity with regular mining procedure and in agreement with the approved schedule shall, in case of disagreement, be settled by arbitration through experts.

Paragraph 17, law of 1917, says:

Any one that upon the enforcement date of this law has already acquired or subsequently by way of a concession acquires allotments or mine pits that come under mining legislation may for a certain period of time have conferred upon him the same privileges as those due to Norwegian national citizens to locate, record, and claim-stake or in some other manner claim allotments or mine pits within one or more definitely established areas.

Paragraph 18 of the same law says:

In instances of trial operations of mining properties, there shall not be a demand for a concession as long as the purpose is none other than a temporary investigation of the extent of an ore deposit, to serve as a basis for calculations regarding the value of a mining property, and in the course of which not more than 1,000 cubic meters of rocky material is extracted in one year from each separate test location (layer, shaft, et cetera). Permission to extract more than 1,000 cubic meters in one year may, however, be granted by the Department of Commerce. Whether the test location does actually serve the intended purpose shall, in cases of doubt, be decided by the department, which shall indicate when such work shall be stopped or reduced. Infringements of the provisions of the department shall be punishable with fines as high as 10,000 kroner (Norwegian crowns[13]).

Duration.-- Concessions shall be granted for a definite period of time, up to 50 years, commencing with the date the concession is authorized and reported. The disposition of mining property at the expiration of this period is covered by paragraph 50 of the law of 1842 and section 13, paragraph 13, and paragraph 15 of the law of 1917.

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13 - The exchange value in U. S. currency of the Norwegian krone in 1931 was 25.0546 cents.

Renewal.-- A renewal of a concession may be granted to cover the rights and conditions of the original concession. (Par. 17, law of 1917.)

Transfer.-- The transfer of an allotment or mine pit to another, even if the transferee is a Norwegian "community resident or a national citizen" necessitates a renewed concession. The transferee must, in any event, conform with all the provisions and requirements of the original concession, as well as with such provisions as may be inserted in the new concession. (Sec. 15, par. 13, law of 1917.)

Obligations and prohibitions of concessionaire.-- The concessionaire shall reimburse, in whole or in part, any expenditures made for the maintenance and repair of public roads, bridges, and piers, whenever such expenditures are traceable to wear and tear caused by mining operations. Roads, bridges, and piers erected by the concessionaire are to be placed at the full disposal of the general public, to the extent that the Department of Commerce decides may be done without embarrassing the mining operations. (Sec. 8, par. 13, law of 1917.)

A concessionaire may not, without the consent of the Department of Commerce, enter into any price-control agreement within the Nation. (Sec. 9, par. 13, law of 1917.)

TAXES, ROYALTIES, AND FEES

Taxes

The circular[14] of the division of commercial laws (Bureau of Foreign and Domestic Commerce) quoted with respect to the incorporation of foreign companies, summarizes the various income and property taxes that must be paid to the Government, national and local. In addition to other taxes, foreign shareholders, whether individuals or corporations, must pay an income tax of 20 per cent on the annual dividends paid by the Norwegian company.

Royalties

A production assessment, under each concession, may be imposed upon each ton of ore. For the first 10 years, the amount shall be set forth in the concession document; for each 10 years following, the amount shall be that agreed upon between the concessionaire and the Department of Commerce. In case of disagreement, the amount shall be decided through the arbitration of three men (appointed by the King in each separate instance), whose decision shall be final, and whose expenditures shall be paid by the concessionaire and the State treasury.

The assessment will amount to a percentage (established by the King) to correspond to a certain value of the ore during the following 10 years, at

14 - Lund, Marquard H., Taxation and Company Law in Norway: Spec. Circ. No. 251, div. com. laws, Bur. For. and Dom. Com., Dec. 11, 1930, p. 3-4.

the point of extraction, ready to be concentrated or exported, after deductions have been made for cost of mining and treatment, including transportation from the pit to the beneficiation plant, but without deductions for administrative and installation expenditures or for interest on capital or for other general outlays. The King may waive for a certain period the payment of the assessment--for instance, when the ore is refined in the country. (Sec. 14, par. 13, law of 1917.)

The yearly amount of the production assessment (dealt with in section 14, paragraph 13, law of 1917) shall be calculated by the Department of Commerce on the basis of the yearly account book of the concessionaire, in which shall be recorded the kinds and the quantities of mineral thrown on the dumps, beneficiated, or reserved for future treatment. The account book must be sent to the department before the expiration of the first quarter of the following year. Failure to keep such a book or to send it in when due is punishable by fines and by imprisonment of not more than four months.

Royalty must be paid within four weeks after the concessionaire has received notification of its amount. If payment is not made, a yearly interest charge of 6 per cent shall be added, or mortgage proceedings shall be instituted. (Par. 14, law of 1917.)

Every newly erected smelting plant or furnace shall, during the first three years of operation, be granted exemption from the production assessment. (Par. 74, law of 1842.)

Fees

The King shall decide the fees payable in connection with applications for the acquisition of mines under the law of December 14, 1917. (Par. 1, law of July 17, 1925.)

MISCELLANEOUS

Management and Supervision of Mining Properties

Paragraphs 42 to 47 of the law of 1842 contain provisions covering such matters as the setting up by the Superintendent of Mines of accurate boundary and landmarks, the supervision by Government officials of mining operations, and the keeping of records and the making of reports by the mine operators.

The Superintendent of Mines shall have the right to demand access to the original mine-plant and production data, in order that he may ascertain the extent of the mine's activities. (Par. 71, law of 1842.)

The concessionaire shall be liable to make reimbursement to the public authorities for expenditures incurred in the way of additional police service. (Sec. 6, par. 13, law of 1917.)

Labor Provisions

Paragraphs 54 to 65 of the law of 1842 cover the duties of the mine employer toward employees, with respect to labor records (par. 54 and 55), shifting of labor when necessary (par. 59), removal notices (par. 60), care of incapacitated workers (par. 61 to 65).

Under no circumstances shall merchandise be forced upon employees or laborers in lieu of cash as compensation for their work. Nor, with the exception of explosives, tools, and other work materials, the laborer shall not be forced to incur any obligations with respect to the purchase of merchandise. (Par. 67, law of 1842; sec. 5, par. 13, law of 1917.)

Sections 5, 6, and 7 of paragraph 13, law of 1917, cover provisions with respect to the employer's obligations toward the temporary and permanent housing of employees, medical service, use of profits from a company store for beneficial purposes, abatement of bootlegging and smuggling of intoxicating liquors, and like matters.

Special Provisions Regarding Financing Mining Operations

Any one advancing money for mining or allied operations or taking financial interest therein (either by furnishing cash or providing equipment) shall enter into a contract, in which shall be enumerated all conditions under which both parties to the agreement shall act. The contract shall state the exact advantages to the one advancing capital; and the contracting parties shall not be bound by the ordinary provisions regulating the payment of interest on money advanced, irrespective of whether collateral has been given. (Par. 52, law of 1842.)

After the contract has been publicly recognized by the local court, the lender shall enjoy a mortgage privilege in the products of the mining property, and he shall become a preferred creditor after the deduction of any fees due the Government treasury. (Par. 52, law of 1842.)

A checking by the lender and the borrower of the mine's account books "ought" to take place twice a year; and until such a check is made or demanded by the lender, no advance demand (on account) shall be legal. (Par. 53, law of 1842.)

Concentration Plants, Et Cetera

The installation of a concentration plant, furnace, or forge to be operated by coal or wood brought from the forest shall first obtain royal consent. (Par. 28, law of 1842.)

I. C. 6657

September, 1932

U. S. BUREAU OF MINES
Bartlesville, Okla.

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

CLAY MINING METHODS AND COSTS AT THE CORUNNA (MICH.) PIT OF THE AETNA PORTLAND CEMENT CO.

BY

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

CLAY MINING METHODS AND COSTS AT THE CORUNNA (MICH.) PIT OF THE AETNA PORTLAND CEMENT CO.[1]

By Oliver A. Dibble[2]

INTRODUCTION

This paper is one of a series being prepared for and published by the United States Bureau of Mines, describing mining methods and costs at clay pits throughout the United States. These papers are designed to disseminate technical information regarding the methods used. The cost tabulations represent operating expenditures only and not total costs. It is recognized that publication of total costs might in many instances cause embarrassment to individual operators as well as to the industry as a whole. On the other hand, operating costs are essential to the technical discussion and study of the methods employed. The attention of the reader is specifically called to this differentiation in order that no misunderstanding of the scope of the cost tabulations shall ensue.

The open-pit recovery of clay for cement plants is a comparatively simple operation in comparison with underground mining of clays; nevertheless, it represents an appreciable factor in the cost of cement manufacture.

The following report deals specifically with the methods employed by the Aetna Portland Cement Co., which operates cement mills at Fenton and Bay City, Mich., in recovering clay from their pit at Corunna, Mich., on the Grand Trunk Western Railway.

HISTORY

In 1903 the Aetna Portland Cement Co. started its cement mill at Fenton, Mich., and at that time opened the Corunna clay pit. Almost simultaneously the Hecla Portland Cement Co. of Bay City, Mich., and the Egyptian Portland Cement Co. of Fenton started operation, opening other portions of the Corunna clay deposit. The clay rights were purchased from the Corunna Coal Co. on the northwest 40 acres of the tract (see fig. 1) by the Aetna Portland Cement Co. and on the northeast 40 acres by the Egyptian Portland Cement Co.

The early digging of clay was interesting because of its crudeness. The stripping was let on contract to a man with a ½-cubic yard, stiff-arm Vulcan shovel. About half an acre was stripped to a depth of 3 feet in two months. The contract for loading clay was let to a group of teamsters, and the digging was done with slip scrapers; these were drawn to a small trestle and dumped into narrow-gage cars, which had been acquired from the Corunna Coal Co. These cars were then pulled by teams up another trestle where they were dumped into flat cars with 10-inch sides and no end gates. These cars held from 15 to 18 tons. The property was operated in this way during the first summer. The next year, however, the slip scrapers were replaced by the previously mentioned Vulcan shovel. This shovel was so handicapped in its

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6657."

2 - One of the consulting engineers, U. S. Bureau of Mines, and general superintendent, Aetna Portland Cement Co.

work because it could not swing, that it was abandoned in the following year, 1905. The procedure then adopted was to use teams with wheel scrapers to dig and haul the clay up a trestle, and to dump the load by hand into railroad cars on the track below. These cars differed from the others only in that they had 24-inch sides. With this plan, 13 wheel-scraper teams and two plow teams could load twelve 30-ton cars daily.

In 1906 the Hecla Portland Cement Co. moved its clay operations to a pit near Bay City, leaving only the Egyptian Portland Cement Co. and Aetna Portland Cement Co. digging at Corunna. These companies continued the slip-scraper plan of digging at a contract price of $0.24 per ton from 1905 to 1916.

In 1916 the Aetna Portland Cement Co. cancelled their digging contracts and purchased a 5/8-yard, full-circle swing, 4-wheel, traction-type shovel fitted with a 12-foot boom. This shovel with the assistance of eight men loaded five 40-ton cars in a 10-hour day. In 1919 the Egyptian Portland Cement Co. bought a 5/8-yard shovel and followed the Aetna plan of digging. In 1923 the Bay City cement mill of the Aetna Portland Cement Co. started operations, which required an increased clay production. About that time the Egyptian Portland Cement Co. purchased a 1-yard full-circle swing, 4-wheel, traction-type shovel fitted with a 20-foot boom and sold the old shovel to the Aetna Portland Cement Co. to care for the extra tonnage required. In 1927 the Egyptian Portland Cement Co. discontinued operations at Fenton and the Aetna Portland Cement Co. bought their 1-yard shovel and all their clay rights. Two years later the Aetna Portland Cement Co. purchased the Rose farm which adjoins its other property on the south (see fig. 1) and in the summer of 1930 started digging from this deposit. In addition to this property the company owns 80 acres containing approximately 2,000,000 cubic yards of high-grade clay near Midland, Mich., on the Pere Marquette Railway. This is to be opened for use at the Bay City Plant in the near future.

PHYSICAL CHARACTERISTICS OF THE DEPOSIT

The Corunna pit comprises approximately 100 acres, of which 80 have been dug over to an average depth of 12 feet. The remaining 20 acres are estimated to contain 380,000 cubic yards of available material, averaging 15 per cent sand.

Those sections of the deposit which were first worked were low in sand. The last deposit (Rose farm), however, contains considerable sand, and consequently the material is more difficult to grind to the required fineness. There is no definite stratification to the clay and on this particular deposit there is practically no overburden. The little overburden is a clayey soil which mixes well with the clay and causes no difficulty. Occasionally pockets of sand and gravel are found in the clay. These are of irregular occurrence and are shovelled to one side as the clay is removed.

SAMPLING

Due to the sand and gravel pockets, accurate sampling of the deposit is difficult and the results are not always representative. If a sand pocket is struck it is bound to contaminate the sample, and it is impossible to determine the area of usable clay without a great many test holes. However, test holes give a fair idea as to the location of available material, and they are put down every 50 feet for each shovel cut. Two men with an auger can sink an average of five holes in 10 hours. The cement-plant laboratories run the samples during spare time. One man can run approximately 20 samples in a 10-hour day.

Sand content is an important factor in clay, due principally to its high resistance to grinding, as fine preliminary grinding is essential to good burning and a fine finishing grind is essential to high quality in Portland cement.

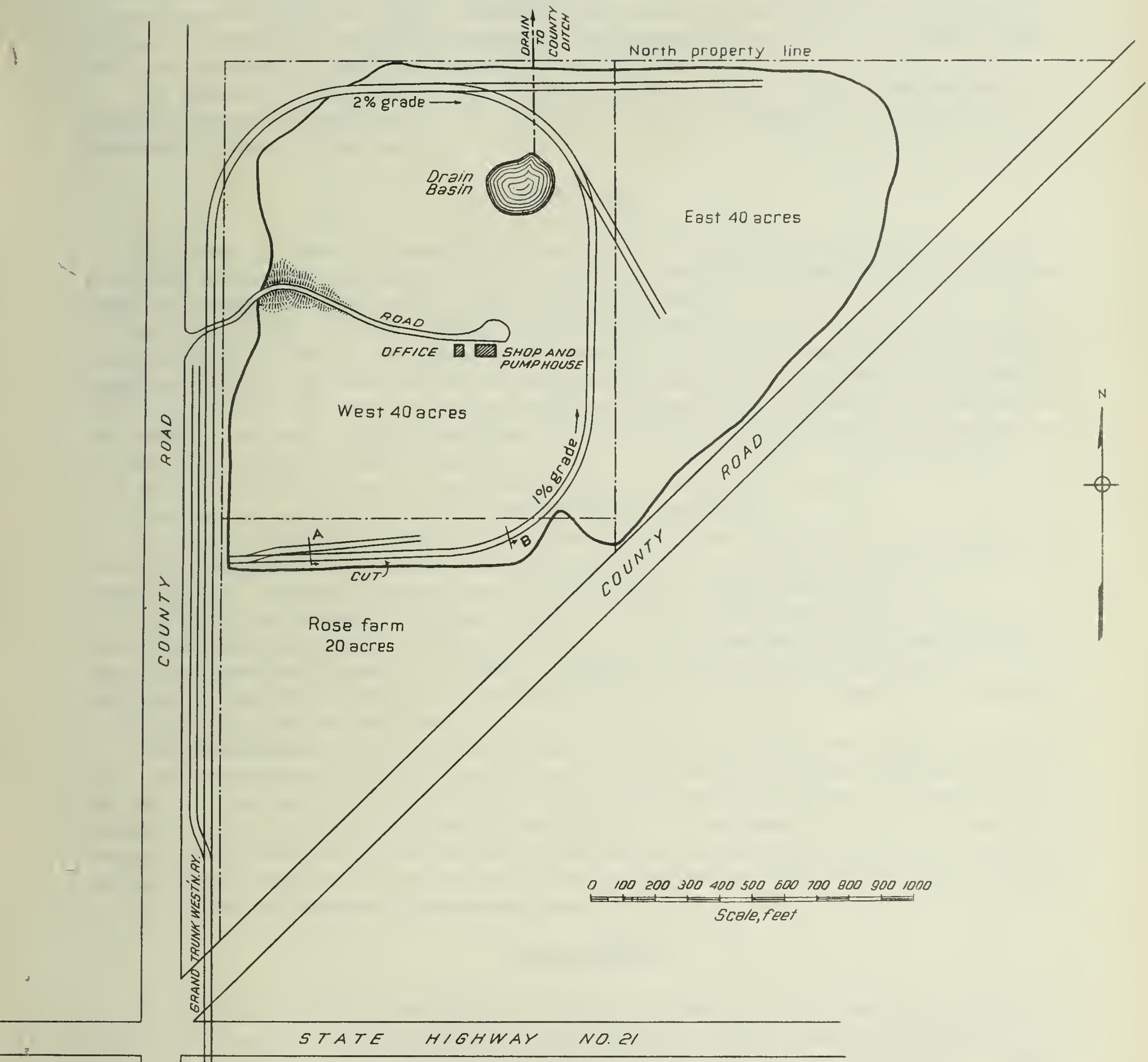

Figure I.- Property map

The determination of sand in the clay is made from samples taken as follows: The auger is sunk about 6 inches at a time and then pulled, and a small sample is taken from each inch of the auger and placed in a bag. Separate bags are used for the material from each 5 feet of hole; if a definite change in the deposit is observed at any time, a new sample bag is used, and the depth at which the change occurred is noted.

The samples are then sent to the laboratory, dried, ground, and a 100-gram sample is washed through a 100-mesh sieve. The oversize is dried and weighed, the weight being the percentage of sand in the clay.

MINING METHOD

The method of mining is simple. The clay is dug in a single bench by two steam shovels which load directly to the railroad cars. However, in year-around operations there are such problems involved as track moving, drainage, and blasting.

The shovel cuts are approximately 50 feet wide and 10 to 20 feet in vertical depth, averaging about 12 feet. The larger shovel takes a full 50-foot cut, while the small one takes about 40 feet, so that when the two shovels meet either the larger shovel continues and finishes the cut or the small shovel doubles back, depending on the demand for material and the length of the cut. The cut now being made is approximately 500 feet long, and the two shovels load from 450 to 500 tons per day.

When a cut is completed all waste material is thrown over the track out of the way, and a tractor and men move the track into position for the next cut.

The pit is worked 10 hours a day with two shovels and an average crew of eight men. The actual digging does not begin until 8:30 or 9 a.m., as steam has to be raised in the shovel boilers. During this delay the men are employed in track maintenance.

The large shovel with the 1-yard dipper can load a 50-ton car in 25 minutes and the smaller shovel with the 5/8-yard dipper can load a car in one hour. In extremely cold weather it is sometimes necessary to blast to loosen the clay ahead of the shovel. In this case 2-inch holes are drilled and light charges of dynamite are used. However, the blasting done during the year is negligible.

The requirements of the two cement mills do not demand the total capacity of both shovels. The present cut is high in sand at one end and low at the other. The Bay City mill can use the clay with the higher sand content but the Fenton mill can not; hence the two shovels are used to dig the different types of clay for the respective mills. A beneficial change could be made by using a 3/4-yard dipper on the 1-yard shovel. The clay is hard to dig due to its cohesiveness, and the use of an over-powered shovel would materially lengthen its life while not appreciably decreasing production.

TRANSPORTATION

The railroad switch engine places the empties and takes out the loaded cars about 7 a.m. If the cut is short the engine switches the empty cars to the siding on the upper grade of the track which is on the south side of the pit (see A, fig. 1), and as cars are required they are switched out with the shovel winches.

Track moving is necessary about every three months. It delays mining and is a costly operation, especially in wet weather, while in dry weather it means a delay of 1½ days with little extra cost.

When a cut is completed all cars are switched out, all waste is thrown out of the way, and the track is cut in two places, at A and B (see fig. 1). A local farmer and his tractor are hired to assist the regular pit crew. The track is pulled into the new position, and

ties and rails laid to connect it with the unmoved track. In dry weather cinders are then tamped around the ties for ballast, but in wet weather the cinders must be laid first and a larger crew is necessary. Moving and preparing the track for temporary use takes about two days of 10 hours each, and an average of about 100-man-hours is required in addition to, bring the track to standard condition. This additional work is done by the regular operating crew during their spare time.

A continual track maintenance is carried on by the regular crew to the extent of about 10 man-hours per day in addition to supervision. Maintenance, of course, requires more time in wet weather than in dry.

MISCELLANEOUS EQUIPMENT

A team is employed continuously for hauling supplies such as coal, etc. There is a general pipe and forge shop for temporary repairs. A ½-hp., 2-cycle gasoline engine with 3-inch bore and 5-inch stroke drives a 3-cylinder, 8-inch duplex pump which delivers 30 gallons of water per minute to the shovels against a 15-foot vertical head. An interesting fact regarding this engine is that it has given seven years service with a repair cost of only $2.45.

Figure 2 shows the organization of the operating personnel.

The wages paid are as follows:

Wage Scale

	Monthly basis	Hourly basis
1 foreman....	$175.00	-
2 engineers	-	$0.50 and $0.45
2 firemen....	-	.40
3 laborers..	-	.40

TABLE 1.- Clay-Pit Costs

Corunna pit. Period, Feb. 15 to Dec. 15, 1931.

Tonnage excavated 112,500.

	Man-hours	Pay roll	Cost per ton
Labor, operation:			
Supervision	2,300	$1,548.00	$0.0138
Engineer	4,320	1,944.00	.0173
Firemen	4,320	1,728.00	.0153
Pitmen	6,040	2,416.00	.0215
	16,980	7,636.00	0.0679
Labor, track moving:			
Supervision	100	67.30	.0006
Labor	920	379.50	.0034
Tractor and driver	150	150.00	.0013
	1,170	596.80	0.0053
Labor, track maintenance:			
Supervision	200	134.70	.0011
Labor	2,600	1,072.50	.0095
	2,800	1,207.20	0.0106
Labor, total	20,950	9,440.00	.0838
Material:			
Coal, tons	150	637.50	.0057
Repairs (1)	-	1,122.30	.0100
		1,759.80	0.0157

Total costs

	Man-hours	Cost	Cost per ton
Labor:			
Operating	16,980	$7,636.00	$0.0679
Track moving	1,170	596.80	.0053
Track maintenance	2,800	1,207.20	.0106
Material:			
Coal	-	637.50	.0057
Repairs	-	1,122.30	.0100
Team:	-	650.00	.0057
Total	20,950	11,849.80	0.1052

Total man-hours per ton 0.1862

(1) Track, ties, and rails, etc., included in this item.

TABLE 2.- Shovel-Operating Costs

Corunna pit. Period, Feb. 15 to Dec. 15, 1931.

Shovel, Type A 1-cubic yard dipper

Material handled: Overburden 5,000 tons
Clay 68,750 tons

	Clay		Overburden		Total cost per
	Amount	Cost per ton	Amount	Cost per ton	ton clay
Engineers	$502.00	$0.0073	$100.00	$0.020	$0.0088
Firemen	449.00	.0065	90.00	.018	.0078
Pitmen	669.00	.0097	74.00	.015	.0108
Foremen	403.00	.0059	80.00	.016	.0070
Fuel	145.00	.0021	30.00	.006	.0025
Repairs	290.00	.0042	60.00	.012	.0051
Total	2,458.00	0.0357	434.00	0.087	0.0420

Shovel, Type O 5/8-cubic yard dipper

Material handled: Overburden 10,000 tons
Clay 43,750 tons

	Clay		Overburden		Total cost per
	Amount	Cost per ton	Amount	Cost per ton	ton clay
Engineers	$274.00	$0.0063	$ 91.00	$0.0091	$0.0083
Firemen	235.00	.0054	78.00	.0078	.0072
Pitmen	337.50	.0077	112.50	.0112	.0103
Foremen	219.50	.0050	73.50	.0074	.0067
Fuel	94.70	.0022	32.80	.0033	.0029
Repairs	159.00	.0036	53.00	.0053	.0048
Total	1,319.70	0.0302	440.80	0.0441	0.0402

Note: The above tables cover cost during actual digging time only, and do not include maintenance labor.

ORGANIZATION CHART

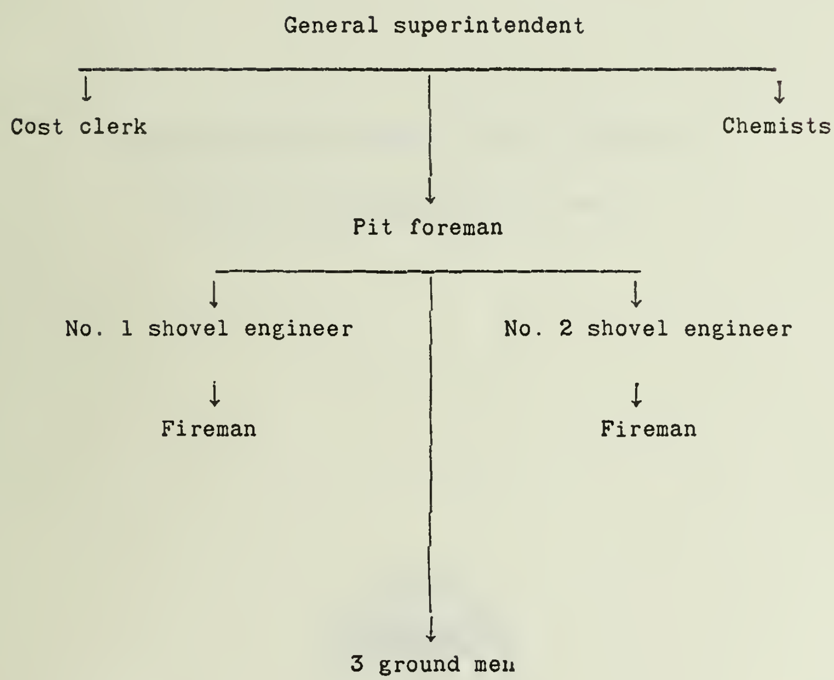

Figure 2.- Organization of operating personnel.

I. C. 6658

October, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MILLING METHODS AND COSTS AT A FLAT RIVER, (MO.) MILL

BY

Will H. Coghill and R. G. O'Meara

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

MILLING METHODS AND COSTS AT A FLAT RIVER (MO.), MILL[1]

By Will H. Coghill[2] and R. G. O'Meara[3]

INTRODUCTION

This paper on milling methods and costs at a flat river (Mo.), mill is one of a series of information circulars on milling methods and costs which is being prepared by the United States Bureau of Mines.

The mill under consideration treats 5,000 tons of lead ore daily by table concentration and flotation. It is 25 years old, and yet it is a new mill. Research has been in continual progress at the plant since its beginning, and accordingly a day never passes without a record of change for advancement. The changes have been so orderly that no evidence of obsolete equipment remains.

The mill is not of the sort to be indorsed by those who stress simplicity in mills; yet the operating cost is phenomenally low. Gravity tailing with a tenor of 0.11 per cent of lead, and flotation tailing with a tenor of 0.15 per cent of lead characterize the metallurgical results.

An unusual policy of the management, reflected by the employees, accounts for the superior accomplishments. "Get the mineral first and afterwards figure the cost of remodeling" is the slogan. Thus the old idea that "the change will cost all the extra mineral is worth" has been rejected.

The mill is very complex. Gravity products have several treatments and some flotation products have many cleanings. But the procedure is so orderly that very little extra expense is incurred.

The mill has 217 motors drawing a load of 4,800 kilowatts. The use of 14 hydraulic classifiers, 116 concentrating tables, and 21 desliming drags is required. Pulp is thickened in two thickeners with an area of 39,000 square feet, and the 8,000 gallons per minute of thickener-water is returned to the mill and circulated through 8 water pumps and 62 sand pumps. Conveying is accomplished by 2,500 feet of conveyor belt and 1,600 feet of elevator belt. The flotation requires a quarter of a mile of pneumatic flotation machines. Consideration of all these things together with the results obtained justifies putting before operators the idea that good milling depends more on good equipment well installed than on mere simplicity.

The United States Bureau of Mines in cooperation with the Missouri School of Mines and Metallurgy, Rolla, Mo., began a study of the milling at Flat River, Mo. (Southeast Missouri lead district), in the early part of 1926.

The major investigation was to compare the already installed distributed-feed system with classification for tabling. Although classification had experienced several unfortunate trials, the first tests justified remodeling one section of a mill for classification. As a

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6658."

2 - Supervising engineer, ore dressing section, metallurgical division, U. S. Bureau of Mines; and supervising engineer U. S. Bureau of Mines, Mississippi Valley Experiment Station, Rolla, Mo.

3 - Assistant metallurgist, U. S. Bureau of Mines, Mississippi Vally Experiment Statio, Rolla, Mo.

result of the improved performance a second section was remodeled, and the change was continued section by section in the first mill and others until finally five of the six mills of the district were remodeled for classification. The alterations of the flow sheets were carefully planned for Sundays so that none of the mills lost time.

Formerly, in order to maintain a satisfactory grade of concentrate and tailing, it had been necessary to wet-screen through 10 mesh. The operation was expensive and generally unsatisfactory.

By the new plan the wet screens were replaced by the classifiers; these sent a sized feed to the tables from which as much coarse material as desired could be sent back to the ball mills and again to the classification system. Thus the ball mills were in closed circuit with classifiers and tables, whereas they were formerly in closed circuit with screens. The new circuit was flexible and made a perfectly prepared feed to the ball mills. In a battery of 10 tables the first 5 could return the tailing to the ball mills while only the last 5 made tailing to waste; the dividing point could be changed at will.

On account of the coarse chats which contaminate the table concentrate the mill under discussion has not been able to take full advantage of the screenless sections adopted in some of the other mills; however, the screens have been reduced in number and are of a coarser mesh than those employed before classification was installed.

Although many improvements were in progress and ultimate results depended upon many factors, it may be said that the metallurgical results shown in Table 28 would not have been attained without classification. The table tailings of the first mill that was remodeled had been the most unsatisfactory of all the mills of the district, but after installing the new system the tailings were markedly the best of the district.

ACKNOWLEDGMENTS

The hearty cooperation of all the mill operators is hereby acknowledged. Special acknowledgment is made to T. J. Clifford, one of the operators, who aided in the preparation of the tabulated material in this report.

Of the Missouri School of Mines and Metallurgy, Horace Scruby gave valuable assistance in the district. Later, A. B. Campbell and J. B. Clemmer, both of the U. S. Bureau of Mines, assisted. Acknowledgment is made to Alexander M. Gow for valuable service in the preparation of the report.

LOCATION AND GENERAL LAYOUT

The mill described in this paper is located at Flat River, Mo., 70 miles south of St. Louis. The mill is housed in a steel-and-concrete building adjacent to the hoisting shaft and is divided into three sections: The primary crushing plant, the secondary and roll crushing plant, and the concentrator. The three sections are connected by enclosed conveyors. The concentrate drying plant, shops, and other buildings are nearby.

The mine is a part of the network of mines supplying about 22,000 tons of ore daily to the several mills of the "Lead Belt" district. Most of the mines are connected by an extensive underground haulage system. Mining is by open stoping with pillar support. Timbering is rarely necessary. The stopes are large, and the ore is loaded by mechanical shovels into 2.5-ton cars, trammed to the shaft pockets, and hoisted in skips of 6.3 tons capacity. All the ore is delivered through a 2-compartment shaft to the crude-ore bin. Underground grizzlies are not used, so that the ore is unsized when it reaches the mill.

The building, which is covered with corrugated iron, is well ventilated and lighted by a large number of swinging windows. It is also well illuminated artifically by electric

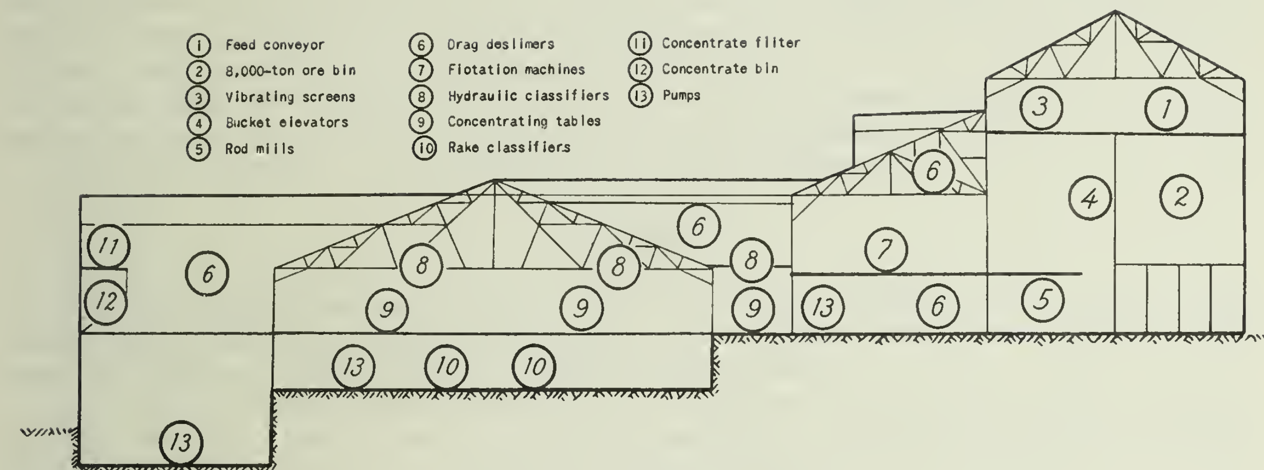


Figure 1.- Diagrammatic section through mill

lights. Although fireproof in construction, the building is protected by an automatic sprinkling system.

The concentrator is built on two terraces and has four principal floors, as indicated in diagrammatic section, Figure 1. On the bottom floor, or first terrace, are belt conveyors and pumps for handling table products and flotation tailings. At a slightly lower elevation are other pumps to raise the secondary slime to the thickeners. On the second floor, which is an extension of the second terrace, are the concentrating tables. Rod mills, drag deslimers, and flotation-product pumps are located on the second terrace. The foundations of the concentrator bin are also on this terrace and the discharge of the bin is only a few feet above. The bin extends to the top of the mill and occupies much of the space above the terrace floor. On the upper floors and intermediate galleries are the flotation machines, screens, drag classifiers, flotation-feed distributors, and hydraulic classifiers.

Although designed for a flow sheet which is now obsolete, the mill was so well planned that it has been adapted to subsequent changes. A notable feature in the present design is the storage facilities. Bins of 180, 900, and 8,000 tons capacity, respectively, are at the heads of the primary breaking unit, the secondary and roll crushing unit, and the concentrator. The large storage bin at the head of the concentrator allows continuous mill operation in spite of short delays in the dry-crushing units.

An abundant water supply is available from the mine, but most of the water is recovered and reused, some within the mill and some through the circulating system.

Power is purchased from the Union Electric Power and Light Co.

ORE TREATED

The only metal of economic importance in the ore is lead; however, minor amounts of iron, copper, zinc, cobalt, nickel, and silver occur. They appear in small amounts in the feed, and are graded up in both the table and flotation middlings.

The lead occurs as galena. The presence of oxidized lead has been suspected but not proved. The iron, copper, and zinc occur as sulphides in mixtures of pyrite, marcasite, chalcopyrite, and sphalerite. Likewise, the cobalt, nickel, and silver probably are present as sulphides.

The gangue is chiefly dolomitic limestone. However, siliceous substances, such as sandstone, and shaly glauconitic and chloritic minerals are present.

The tenor of the ore is about 3.5 per cent of lead, occurring as galena disseminated through the gangue. Jackson[4] writes of the character of the ore as follows:

> In general, it may be said of the district that galena occurs disseminated through dolomitic limestone and shaly beds, in horizontal sheets and along bedding planes, in vugs and cavities, filling or lining walls of joints and crevices and as aggregates of cubes in channels and joints.

A chemical analysis of the ore is given in Table 1.

The dolomitic limestone is a hard, dense variety fairly resistant to crushing. The glauconitic and chloritic material is largely eliminated in the primary slime. The sandstone is present in such small amounts that its resistance to grinding is negligible. Galena breaks readily, but the other sulphides are more difficult to crush.

- Jackson, C. F., Methods of Mining Disseminated Lead Ore at a Mine in the Southeast Missouri District: Information Circular 6170, Bureau of Mines, Sept., 1929, p. 3.

Table 1.- Chemical analysis of the ore

Constituent	Assay, Per cent
Lead	3.5
Silica	3.2
Alumina	1.3
Iron oxide	6.6
Lime	27.4
Magnesia	15.5
Manganese oxide	.7
Sulphur	.8
Copper, nickel, cobalt	.1
Zinc	.2
Carbon dioxide (by difference and undetermined)	40.7
	100.0

Some of the lead is finely disseminated and some is more coarsely crystalline. Formerly, when a 5-mesh feed was sent to the tables the coarsest concentrate was too contaminated with locked gangue to be marketable. Now, a feed through 7 mesh gives a product of acceptable grade. Locking is also noticed in the gravity tailing, the lead content of which is never in a free state.

None of the gangue is entirely free from lead. The specific gravity of particles of crushed ore ranges from less than 2.85 that of galena (7.5), and all specific-gravity increments of any mesh contain lead. The assay per cent of lead in the increments of stated sizes and densities is given in Table 2. The lead assay of each specific gravity increment, excepting the sink in 3.34 specific gravity, decreases as the finer sizes are approached.

Table 2.- Assay of percentage of lead in the specific-gravity increments of the ore

Size, mesh	Assay, lead, per cent				
	Float on 2.85 specific gravity	2.85 to 2.90 specific gravity	2.90 to 2.95 specific gravity	2.95 to 3.34 specific gravity	Sink in 3.34 specific gravity
Plus: 4	0.36	0.35	1.07	7.74	19.10
6	.18	.15	1.07	7.64	45.46
8	.16	.14	.76	6.86	54.93
10	.16	.09	.59	6.74	59.32
14	.09	.09	.36	5.98	60.50
20	.07	.08	.25	5.06	60.84
28	.07	.07	.25	4.90	64.39
35	.06	.06	.14	4.68	64.05
48	.05	.05	.13	4.60	61.52
65	.05	.05	.09	3.30	64.39
100	.05	.05	.08	2.50	65.57

Other sulphides as well as lead are locked. The lightest increment in Table 2, the float on 2.85 specific gravity, contains a higher percentage of lead than the corresponding sizes of the next heavier increment, 2.85 to 2.90 specific gravity. This lightest portion

is the most siliceous part of the ore and the locking with lead is more persistent; finer crushing is required to liberate it.

The siliceous character and the locking of lead in the float on 2.85 specific gravity is present in even a lighter increment. The separation between 2.80 and 2.85 specific gravity, showing the same persistent locking, is given in Table 3.

If the mill were making a tailing with a tenor of 0.75 per cent of lead, the locking mentioned would not come up for consideration, but when a tailing of 0.11 per cent of lead is made, as is the practice at this mill, the dissemination is very important.

Table 3.- Persistent locking of lead, and siliceous character of lightest specific-gravity increments

Product	Weight, per cent	Assay, per cent Lead	Assay, per cent Insoluble
Float on 2.80 specific gravity	12.30	0.12	30.21
2.80 to 2.85 specific gravity..	55.28	.01	5.92
Sink in 2.85 specific gravity..	32.42	-	-
Total........	100.00	-	-

The entire story of the dissemination of lead is not told by the assays of the increments alone. The weight percentages and distribution of the lead add to the knowledge of the character of the ore. This information is given in Table 4.

The sink in 3.34 specific gravity is not indicative of the grade of concentrates that can be made. The galena itself is high grade; hand-picked coarse galena and fine table concentrates assay as high as 84 per cent of lead. As previously mentioned, considerable free galena is liberated from the coarse sizes. The ultimate freeing of lead, however, requires fine grinding.

Thus far the discussion of locking has related primarily to the galena and nonsulphide gangue. Galena is also locked with other sulphides. This was shown by a microscopic examination of a flotation middling, where it was observed that the iron and copper content was high and the locking of the lead with sulphides was more pronounced than with dolomite.

HISTORY

The mill was built in 1906, about a decade before flotation was introduced into the district; and gravity concentration was predominant in the original flow sheet. The mill was the largest in the district and had a capacity of 2,600 tons daily. A feature was a separate outside crushing plant where the ore was dry crushed to pass 8-millimeter screens. It then passed through a sampling plant to the mill storage bins. This arrangement was a big advance over anything practiced in the district. Three screen sizes above the 2-millimeter size were made and each was treated on Harz jigs while the undersize of the 2-millimeter screen was classified. Some of the spigots were jigged and others were tabled.

Later the Harz jigs were replaced by Hancock jigs. This change simplified the flow sheet and enabled the capacity to be increased from 2,600 to 4,000 tons per day.

Table 4.- Specific-gravity analysis of mill feed, per cent

Size, mesh	Float on 2.85 specific gravity			2.85 to 2.90 specific gravity			2.90 to 2.95 specific gravity			2.95 to 3.34 specific gravity			Sink in 3.34 specific gravity			Total		
	Weight	Assay, lead	Distribution of lead	Weight	Assay, lead	Distribution of lead	Weight	Assay, lead	Distribution of lead	Weight	Assay, lead	Distribution of lead	Weight	Assay, lead	Distribution of lead	Weight	Assay, lead	Distribution of lead
Plus: 4	7.47	0.36	0.57	1 07	0.35	0.08	0.48	1.07	0.11	0.60	7.74	0.86	0.24	19.10	0.95	9.86	1.23	2.57
6	15.31	.18	.58	2.09	15	.07	.90	1.07	.20	1.24	7.64	2.00	.70	45.46	6.80	20.24	2 25	9.65
8	10.79	.16	.37	1.62	.14	.05	.61	.76	10	.71	6.86	1.03	.76	54.93	8.83	14.49	3.38	10.38
10	7.30	.16	.25	2.12	.09	.04	.45	.59	.06	.48	6.74	. 68	.83	59.32	10.49	11.18	4.86	11.52
14	5.51	.09	.10	1.29	.09	.03	.40	.36	.03	.30	5.98	.38	.70	60.50	8.98	8.20	5.48	9.52
20	3.29	.07	.05	1.10	.08	.02	.43	.25	.02	.15	5.06	.16	.46	60.84	5.97	5.43	5.41	6.22
28	2.43	.07	.04	.94	.07	.01	38	.25	.02	.10	4.90	.10	.51	64.39	6.95	4.36	7 71	7.12
35	1 79	.06	02	.86	.06	.01	.26	.14	.01	.06	4 68	06	.43	64.05	5.81	3 40	8.20	5.91
48	1.64	.05	.02	.81	.05	.01	30	.13	.01	05	4.60	.05	.38	61.52	4.88	3 18	7.38	4.97
65	.93	.05	01	.79	05	.01	.23	.09	.00	.03	3.30	.02	.31	64.39	4.28	2.29	8.91	4.32
100	.79	.05	.01	.95	.05	.01	.25	.08	.00	.03	2.50	.02	.35	65.57	4.89	2.37	9.82	4.93
Minus: 100	-	-	-	-	-	-	-	-	-	-	-	-	-	-	[1]22.89	15.00	7.20	22.89
Total......	57.25	0.17	2.02	13.64	0.11	0.34	4.69	0.57	0 56	3.75	6.92	5.36	5.67	57.21	91.72	100 00	4.72	100.00

1 - All the lead in the minus 100-mesh product is assumed to be free.

The size of the mill feed was increased from 8 to 10 millimeters, and the plus $1\frac{1}{4}$-millimeter part of the feed was treated on the Hancock jigs to give concentrate, tailing, high-grade middling, and low-grade middling. Additional concentrate was recovered from the high-grade middling by screening on 2-millimeter mesh and taking the oversize as concentrate. The undersize was reconcentrated on a jig from which a tailing, a middling, and a concentrate were obtained; the tailing was tabled to remove a small amount of fine lead, and the middling containing zinc, lead, iron, and copper was treated in a magnetic plant along with the table middling. The mixed sulphide middling was roasted and the pyrite and chalcopyrite were removed by a Cleveland-Knowles magnetic machine. The tailing from the magnetic machine -- the zinc concentrate -- was tabled to separate the remaining lead. The magnetic concentrate had 7 per cent of copper and some silver.

The low-grade middling was reconcentrated on jigs. Products similar to those of the "ore" jigs were made. The low-grade middling was reground in rolls and the plus $1\frac{1}{2}$-millimeter part returned to the jigs.

The undersize of the original feed (minus $1\frac{1}{4}$ mm.) and the undersize of the reground jig middling (minus $1\frac{1}{2}$ mm.) were deslimed, classified in hydraulic classifiers, and tabled. The classifiers used were largely of the Richards vortex type, but some were of the Bunker Hill & Sullivan type. A tailing, middling, and concentrate were made on all the tables. The middling was reground in rolls.

The slime was treated in spitzkasten and the spigot products were concentrated on tables and vanners. The table and vanner tailings and slimes were treated in a canvas plant. The concentrate of the canvas plant was cleaned on tables and vanners. A lead concentrate was all that was recovered in the slime treatment.

To recapitulate: Before the advent of flotation the concentration was by jigs, tables, vanners, and a canvas plant; roasting and magnetic separation were applied to the complex sulphide middlings.

During the course of this early milling a large tailing pile accumulated, many parts of which had a tenor in lead as high as 1 per cent.

In 1914 flotation was adopted. This was the turning point in the milling, for slimes could then be treated efficiently. The way was paved for finer grinding and the consequent lower tailing of to-day. Although all-tabling of a minus 2-millimeter feed had been attempted at an earlier date, it had failed because of the inability to recover the lead in the slime.

The first flotation practice consisted in thickening the slime to about 20 per cent of solids and floating the thickened product. Crude wood creosote was used as the frothing agent; acid was not used.

Since 1914 fine grinding, table concentration, and flotation all have advanced. The dry-crushing plant was remodeled in 1926; the primary breakers were replaced by new crushers and placed in a separate plant. This change made finer grinding possible and increased the capacity to 5,000 tons per day. In 1927 the last jig was discarded.

In 1928, as a result of a cooperative investigation with the United States Bureau of Mines and the Missouri School of Mines and Metallurgy, hydraulic classification was substituted for the "distributed" table feed. Earlier attempts to improve results by classifying the table feed had failed. Whereas desliming had always been practiced, it was not until this change was made that the benefit of actual classification was fully realized.

During the last four years, five mills of the district with an aggregate capacity of 19,000 tons per day have been remodeled to adapt them to multiple-spigot hydraulic classification in place of the customary distributed table feed.

In the last few years low-pressure air flotation machines have almost entirely replaced the mechanical and high-pressure air types. Pine oil or cresylic acid and xanthate are now

generally used as flotation reagents. Trommels have been replaced by vibrating screens. Rod mills have superseded rolls for regrinding middlings. Table tailings that were formerly dewatered and carried to waste by a long series of belt conveyors are now deslimed, mixed with part of the flotation tailings, and pumped to waste. Table concentrates are filtered before shipment instead of merely being dewatered by drags.

PRESENT MILLING PRACTICE

The descriptions which follow are of the milling practice as in 1931. Concentration is by tabling and flotation. Table concentrates at present constitute about 52 per cent of the output and have a content of 76 per cent of lead. On account of much regrinding about two-thirds of the mill feed is eventually treated by flotation to produce the other 48 per cent of the total concentrates. Flotation concentrates are not as rich as gravity concentrates and average 71 per cent of lead.

CRUSHING AND ROLL GRINDING

Dry crushing, in three steps, reduces the run-of-mine ore to 5-mesh (0.17-inch). The first reduction is by breaking in 9E Telsmith gyratories to less than 3 inches, the second by crushing in 48-inch Symons horizontal disk crushers to less than 1.5 inches, and the third step by grinding in 48 by 24 inch St. Joe rolls to pass 5-mesh screens. This size is considered the practical minimum without sacrificing tonnage. The dry-crushing flow sheet is shown in Figure 2.

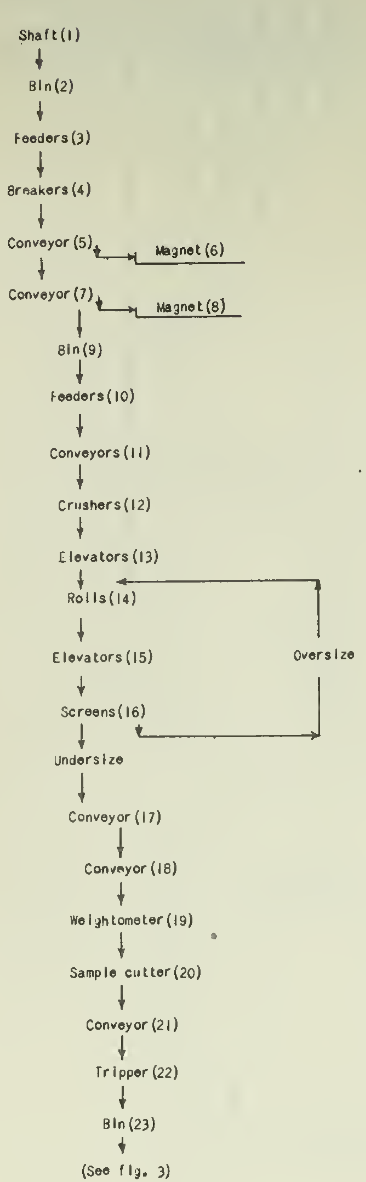
Crude ore is dumped into bins as hoisted. Iron bars hanging in the bin act as buffers and protect the bin wall. Ore is fed to the breakers by apron feeders under hand-operated arc gates and heavy iron fingers. The gates are opened wide and the fingers retard the flow of ore onto the feeders. When the ore hangs up in the bin it is barred down by hand. Grizzlies are not used underground and pieces too large for the breaker are sledged or bulldozed in the primary-crushing plant. A 40-ton crane above the gyratories facilitates repairs and replacements.

The breakers handle in two shifts enough tonnage to employ the disk crushers and rolls for three shifts. The respective capacities in terms of original feed are about 107 tons per hour for each breaker, 56 tons per hour for each disk crusher, and 56 tons per hour for each set of rolls. The 9E Telsmith primary breakers are able to handle a considerably larger tonnage than they do now, for even in their present operation of only two shifts they are not heavily loaded.

Although considered an old type of crushing equipment, the Symons horizontal disks, which have been in use for over 20 years, are retained because they are still able to give the required service. However, it is likely that before many years they will be replaced by newer machinery.

The 48 by 24 inch St. Joe rolls are operated in closed circuit with vibrating screens with 5-mesh openings. This fine roll-grinding necessitates dressing the roll shells about every three months to remove flanges and corrugations. To do this the roll on its shaft is lifted out of the frame and placed in special bearings. It is turned down by rotating at about 2 r.p.m. against a cutting tool that moves automatically across the face. Sometimes a grinding wheel instead of the cutter is mounted on the tool carriage.

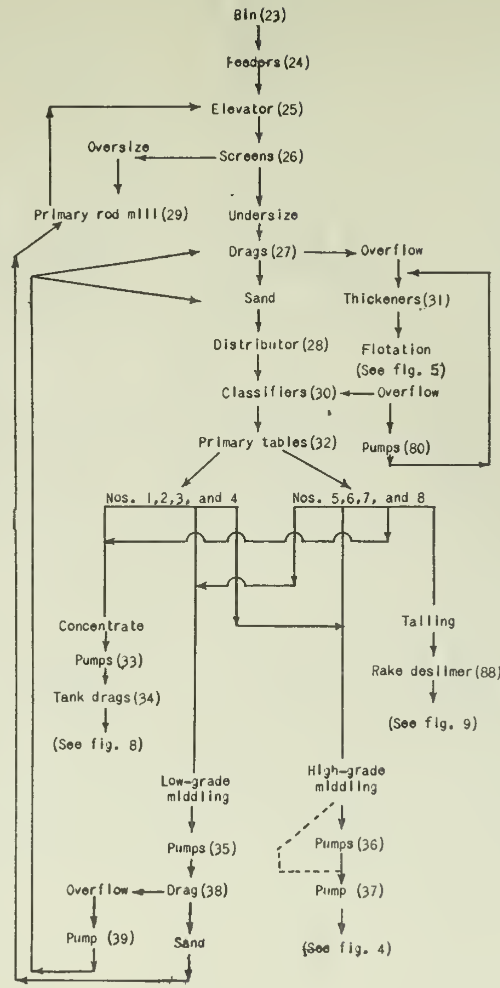
The operating details of the primary crushers, the secondary crushers, and the rolls are given in Table 5. The sizing analyses of the products of each step are shown in Table 6.



Metallurgical details

(1) Two 6.3-ton skips
(2) 180-ton crude-ore bin
(3) Three drum feeders, 4 feet 9 inches in diameter by 4 foot 8 inch face, 0.3 to 0.4 r.p.m., No. 4 Reeves and 5-hp. motor on each
(4) Three 9-E Telsmith primary breakers (see Table 5)
(5) 36-inch belt conveyor (see Table 21)
(6) 36-inch Ohio mill-type electromagnet
(7) 36-inch inclined belt conveyor (see Table 21)
(8) 36-inch Cutler-Hammer electromagnet
(9) 900-ton bin
(10) Two Stephens-Adamson 36-inch pan feeders 8 feet long, 10 feet per minute, No. 4 Reeves and 5-hp. motor on each
(11) Two 24-inch inclined belt conveyors (see Table 21)
(12) Four 48-inch Symons horizontal disk crushers (see Table 5)
(13) Two 28-inch bucket elevators (see Table 22)
(14) Four 48 by 24 inch St. Joe rolls (see Table 5)
(15) Four 28-inch bucket elevators (see Table 22)
(16) Twelve Leahy dry screens (see Table 19)
(17) 24-inch belt conveyor (see Table 21)
(18) 24-inch inclined belt conveyor (see Table 21)
(19) Merrick weightometer, model E
(20) Vezin-type sample cutter, 14.5 r.p.m.
(21) 24-inch belt conveyor (see Table 21)
(22) Jeffrey automatic 2-way tripper
(23) 8,000-ton concentrator bin

Figure 2.- Dry-crushing flow sheet



Mechanical details

(23) 8,000-ton concentrator bin
(24) Three 24-inch belt feeders, 10 feet long; 13 feet per minute; slope, 2.4 inches per foot; fed by hopper with adjustable gate; drive, No. 3 Reeves, DeLaval worm gear, 10-hp., 870-r.p.m. motor. One 24-inch pick-up conveyor, 19 to 38 feet long; 105 feet per minute; drive, bevel gear from feeder
(25) 20-inch bucket elevator (see Table 22)
(26) Two Leahy wet screens (see Table 19)
(27) Two drag deslimers (see Table 20)
(28) 2-way revolving distributor, 15 r.p.m.
(29) 6 1/2 by 12 foot rod mill (see Table 9)
(30) Two hydraulic classifiers (see Table 11)
(31) Two 150-foot Dorr thickeners (see Table 18)
(32) 16 concentrating tables (see Table 13)
(33) 4-inch Morris pumps (see Table 23)
(34) Two tank drags (see Tables 20 and 25)
(35) 4-inch Wilfley pumps (see Table 23)
(36) Morris pumps (see Table 23)
(37) Wilfley pump (see Table 23)
(38) Drag deslimer (see Table 20)
(39) 8-inch Wilfley pump (see Table 27)
(80) 8-inch Wilfley pumps (see Table 23)
(88) Reciprocating rake deslimer (see Table 26 and fig. 9)

Figure 3.- Flow sheet of one section of primary wet grinding and gravity concentration

Table 5.- Details of dry-crushing units

Data	Dry-crushing equipmnt		
	Primary crushers	Secondary crushers	Rolls
Flow-sheet serial	(4)	(12)	(14)
Crushers, in operation	3	4	4
Type	Gyratory	Symons horizontal-disk	Spring type
Manufacture	Smith Engineering Works	Nordberg Engineering Co.	St. Joseph Lead Co.
Size	9 E	48-inch	48 by 24 inch
Opening:			
Top. inches	19½	-	-
Maximum discharge do.	2 to 2¼	7/8	Set close
Minimum do. do.	1-1/16 to 1½	½	-
Gyrations per minute	108	-	-
Eccentric throw inches	One at 5/8 Two at 3/4	-	-
Speed:			
Countershaft r.p.m.	324	-	-
Disk do.	-	100	-
Eccentric do.	-	250	-
Rolls do.	-	-	60.0 to 65.2
Motors horsepower	Three at 150	Two at 250[1]	Two at 250[1]. Two at 150
Circuit, open or closed	Open	Open	Closed
Power consumption, each horsepower	44 average	40 to 45	161 to 164
Material:			
crushing-surface	Manganese steel	Manganese steel	Chrome steel
Frame	Cast steel	Cast iron	Cast steel
Wearing part, life years	-	Disks, 2	Shell, 1
Shifts operated	Two, 8-hour	Three, 8-hour	Three, 8-hour
Oil used per shift per machine gallons	4	4	-
Feed:			
Character	Run-of-mine	Primary crusher discharge	Symons discharge and screen oversize
How fed	Drum feeder	Chute	Launder
New, per hour tons	107	56	56
New, per hp.-hour do.	2.43	1.31	0.34
Total, per hour do.	-	-	309.2
Total, per hp.-hour do.	-	-	1.9

1 - 250-hp. motor for two Symons and one set of rolls.

Table 6.- Sizing analyses of successive dry-crusher products

Size	Weight, per cent				
	Primary breaker discharge (Symons horizontal disk-crusher feed)	Symons horizontal disk-crusher discharge (new feed to roll)	Composite roll feed (new feed and dry screen oversize)	Roll discharge	Finished roll discharge (screen undersize)
Plus: 3.0 inch	11.7	-	-	-	-
2.1 do.	19.0	-	-	-	-
1.5 do.	17.6	13.3	1.8	-	-
1.05 do.	12.0	32.2	4.3	-	-
.742 do.	8.4	23.4	5.0	1.2	-
.525 do.	6.6	10.9	6.9	7.7	-
.371 do.	5.3	7.3	20.9	15.8	-
3 mesh	4.2	4.0	19.3	19.8	-
4 do.	3.7	2.5	19.2	19.5	-
6 do.	2.2	1.0	11.4	11.9	3.0
8 do.	2.0	1.0	6.5	8.5	14.5
10 do.	1.8	.7	1.9	2.3	18.6
14 do.	.9	.5	.6	2.3	12.9
20 do.	.8	.4	.2	1.2	7.9
28 do.	.6	.3	.2	1.2	8.0
35 do.	.5	.3	.1	.8	5.5
48 do.	.5	.2	.1	.6	3.9
65 do.	.4	.2	.1	.6	3.7
100 do.	.2	.2	.1	.4	2.3
150 do.	.2	.1	.1	.4	2.0
200 do.	.4	.2	.1	.3	2.0
Minus: 200 do.	1.0	1.3	1.2	5.5	15.7
Total...........	100.0	100.0	100.0	100.0	100.0

After passing the 5-mesh screens the ore is conveyed over a Merrick weightometer to the concentrator bin. In case the conveyor belts stop, automatic lights signal the operator in the primary-crushing plant to close the crude-ore bin gates.

WET GRINDING AND GRAVITY CONCENTRATION

For discussion, the wet-grinding and gravity concentration are divided into primary and secondary circuits. In the primary gravity circuit, wet-grinding, hydraulic classification, and table concentration are done in six similar sections of the concentrator. One of these sections, the flow sheet of which is given in Figure 3, will be discussed. It deals with the new feed and the returned low-grade middling. The secondary gravity circuit deals with the high-grade middling from the first operation, and will be discussed later. In all, gravity concentration uses 116 tables and 7 rod mills. Table 28 shows that the grade of the table tailing is 0.11 per cent of lead. Such a low tailing would not be obtainable were not every effort made to keep lubricating oil out of the ore.

Primary Gravity Circuit

The ore for each circuit is drawn from the concentrator bin by three belt feeders, is lifted to the top of the mill in a bucket elevator, and is sized by two 7-mesh vibrating screens. On account of the fineness of the dry feed the screens are lightly loaded. The oversize is ground in a primary rod mill and is returned to the screens by the new-feed elevator. The screen undersize is deslimed in two drag deslimers. The overflow runs to flotation thickeners. The sand discharge is divided by a 2-way mechanical distributor and is sized in two 10-spigot hydraulic classifiers. The overflow of the hydraulic classifiers also goes to the flotation thickeners. The 10 spigot discharges of each classifier are treated on eight concentrating tables. The first six spigots feed one table each, while the seventh and eighth spigots, and the ninth and tenth spigots, are combined to feed the other two tables. A concentrate, a high-grade middling, and a low-grade middling are made on each of the first four tables. The last four tables make similar products and a tailing in addition. Sizing analyses of the table tailings and concentrates of the combined primary and middling circuits are shown in Table 7. The low-grade middlings from all eight primary tables are dewatered in a drag deslimer and reground in the primary rod mill. The sizing analysis of the low-grade table middlings is given in Table 8. The high-grade table middlings constitute the feed for the secondary gravity circuit and will be discussed under that caption later.

Table 7.- Sizing analyses of composite table tailing and composite table concentrate

Size, mesh	Weight, per cent	
	Table tailing	Table concentrate
Plus: 14	-	2.1
20	-	3.5
28	3.5	4.6
35	13.1	8.3
48	28.7	14.3
65	42.3	24.0
100	8.6	12.5
150	3.0	12.8
200	.5	8.1
Minus: 200	.3	9.8
Total....	100.0	100.0

Table 8.- Sizing analyses of primary rod-mill products

Size, mesh	Weight, per cent			
	Screen oversize (1)	Low-grade middling drag discharge (2)	Rod-mill feed (1) and (2)	Rod-mill discharge
Plus: 6	2.8	-	0.5	-
8	28.8	0.3	7.8	-
10	36.6	4.6	16.0	-
14	13.6	9.1	14.2	0.2
20	3.7	10.1	8.8	.3
28	2.9	17.1	19.2	10.7
35	1.3	22.0	16.2	17.7
48	1.2	19.4	7.6	15.5
65	1.1	13.4	3.4	14.5
100	.8	1.9	1.1	5.2
150	.7	.6	.7	4.9
200	.7	.2	.5	3.8
Minus: 200	5.8	.5	4.0	27.2
Total....	100.0	100.0	100.0	100.0

Rod-Mill Grinding

The primary rod mill reduces the wet-screen oversize and the low-grade middlings from the primary tables to 20-mesh. In other words, the rod mills are operated in closed circuit with both screens and tables. The new feed to each of the six sections is 850 tons per 24 hours. Because of the circulating load the tonnage of rod-mill feed is about the same as that of the new feed.

The pulp density in the mills is about 40 per cent of solids by volume, 65 per cent by weight. The sizing analyses of the several products are given in Table 8. The mechanical details of the rod mills are given in Table 9. The driving gears on the rod mills may be reversed to compensate for wear.

Unusually long liner lifer is obtained in the rod mills. After five years of continuous service, the estimate has been made that the life will reach seven years. This may be attributed to the softness of the ore, the fineness of the feed, and the method of installing liners. The liner sections are bolted in place with 1/8-inch shims between the liners and the shell of the mill. Then molten zinc is poured between and under the sections so as to form a continuous and cushioned all-metal lining in the mill. After five years it has been found expedient to reverse the sections and rezinc them, because of the breaking loose and erosion of the zinc. The greatest wear on the liners had taken place toward the discharge end of the mill, but rod wear was uniform throughout the length of the rods.

Table 9.- Rod-mill data

	Primary	Secondary
Flow-sheet serial	(29)	(53)
Mills, in operation	6	1
Manufacture	Allis-Chalmers	Allis-Chalmers
Size ... feet	$6\frac{1}{2}$ by 12	5 by 9
Feed:		
Character	Wet screen oversize and table return	Low-grade middlings
Rate per 24 hours ... tons	850	125
Solids by weight ... per cent	66	66
Feeder type	Scoop	Scoop
Discharge type	Overflow	Overflow
Liners:		
Type	Wave	Wave
Weight ... pounds	-	19,660
Composition	Manganese steel	Manganese steel
How held	Bolted and zincked in	Bolted and zincked in
Life ... years	7[1]	6[1]
Zinc used ... pounds	4,800 to 5,600	3,900
Make-up rods:		
Diameter ... inches	2	2
Load ... pounds	55,000	28,000
Consumption ... do.	0.38, per ton of crude ore	2.97, per ton of actual feed
Drive, type	Direct with herring-bone gear	Direct with herring-bone gear
Mill speed ... r.p.m.	16.9	20.8
Motor:		
Type	Induction	Induction
Power consumption ... horsepower	190	95
Capacity ... do.	200	100

1 - Estimated.

The rods are hot-sawed and machine-straightened. The length is 11 feet $11\frac{1}{2}$ inches maximum, and 11 feet $10\frac{1}{2}$ inches minimum. The chemical specification is as follows:

Constituent	Per cent
C	0.60 to 0.75
Cr	0.30 to 0.50
Mn	0.50 to 0.70
Si	0.15 to 0.25
S (max.)	0.04
P (max.)	0.04

The rod charge in the primary rod mills is given in Table 10. The original charge to the primary mills was 55,000 pounds with a maximum size of 4 inches. This maximum has been reduced to 2 inches, and the tendency is toward a further decrease. This is prompted, at least in part, by the thought that the large rods break the small ones prematurely.

Table 10.-Primary rod-mill charge

Rods, inches diameter	After about four years operation			As returned to mill		
	Rods,	Weight		Rods,	Weight	
	number	pounds	per cent	number	pounds	per cent
2-1/2	-	-	-	80	15,912	28.1
2	47	5,983	11.8	38	4,837	8.5
1-7/8	88	9,847	19.5	92	10,295	18.1
1-3/4	135	13,163	26.0	117	11,408	20.1
1-5/8	93	7,812	15.4	75	6,300	11.1
1-1/2	67	4,797	9.5	54	3,866	6.8
1-3/8	84	5,057	10.0	69	4,154	7.3
1-1/4	41	2,038	4.0	-	-	-
1	(1)	1,910	3.8	-	-	-
Total..	555	50,607	100.0	525	56,772	100.0

1 - Broken rods, 2 to 7 feet long.

Hydraulic Classification

Two 10-spigot hydraulic classifiers of the constriction-plate type are used in each of the six primary sections. Two others are in the middling sections. Each is fed about 430 tons per 24 hours by a drag deslimer. The mechanical details of the classifiers are given in Table 11 and some data on the products of one are shown in Table 12. The results of an efficiency test on a similar classifier, giving more complete data, are presented in a recent publication.[5]

Several kinds of spigot liners including rubber and case-hardened steel types, have been tried, but porcelain has been adopted finally. The present liners are porcelain cylindrical tubes about 1-1/8 inches outside diameter and 1 inch long. The holes are 3/4, 5/8, 1/2, or 7/16 inch in diameter, the larger holes being at the feed ends of the classifiers.

5 - Coghill, Will H., Classification and Tabling of Difficult Ores, with Particular Attention to Fluorspar: Paper 453, Bureau of Mines, 1929, p. 38.

Table 11.-Details of hydraulic classifiers

Flow-sheet serial	(30)	(41)	(47)
Classifiers in operation	12	1	1
Cells:			
Number		10	
Size ... inches		10 by 10	
Dividers, height ... do.		12	
Overflow type		Side and end	
Overflow height ... inches		20	
Material		Cypress	
Hole spacing in plates		Corner of 2-inch squares	
Hole diameters ... inches		4 cells, 1/4; 4 cells, 3/16; 2 cells, 1/8	
Discharge type		Continuous spigots	
Spigot diameters ... inches	Two, 3/4 four, 5/8 four, 7/16	Two, 3/4 four, 5/8 two, 1/2 two, 7/16	Two, 5/8 six, 1/2 two, 7/16
Feed:			
Rate per 24 hours ... tons		430	
Pulp density ... per cent		38.5	
Hydraulic water per minute ... gallons		275	
Ratio ... water : solids		3.8 : 1	

Table 12.-Sizing analyses and tonnages of hydraulic-classifier products

Size, mesh	Weight, per cent									
	Feed	Spigot No. 1	Spigot No. 2	Spigot No. 3	Spigot No. 4	Spigot No. 5	Spigot No. 6	Spigots No. 7 and No. 8	Spigots No. 9 and No. 10	Overflow
Plus: 8	0.2	1.6	0.4	0.1	-	-	-	-	-	-
10	3.7	17.8	9.8	2.7	0.3	0.1	-	-	-	-
14	8.0	28.0	19.7	9.3	2.3	1.5	-	-	-	-
20	6.6	17.9	16.4	11.6	4.9	4.1	0.4	0.3	0.2	-
28	11.1	16.0	18.5	20.2	13.1	11.5	3.5	2.6	2.2	-
35	13.2	9.6	15.1	22.0	22.3	16.9	8.8	7.4	6.6	-
48	14.3	4.4	9.5	16.2	24.5	19.8	21.4	18.0	16.1	-
65	20.3	3.2	7.0	12.2	22.9	34.1	41.7	45.0	38.0	-
100	7.4	.8	1.9	3.1	5.0	6.7	13.1	14.8	21.4	1.2
150	5.1	.4	.9	1.4	2.5	3.0	7.0	8.0	12.3	14.3
200	2.3	.1	.4	.5	.9	.3	.4	1.2	1.1	17.1
Minus: 200	7.8	.2	.4	.7	1.3	2.0	3.7	2.7	2.1	67.4
Total	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Mean mesh, millimeters	0.305	0.828	0.623	0.451	0.323	0.287	0.215	0.207	0.195	-
Tons per 24 hours	[1]418.4	52.1	78.4	41.4	48.8	49.4	47.6	52.3	48.4	-

1 - This tonnage does not include the slime in the overflow.

Table Concentration

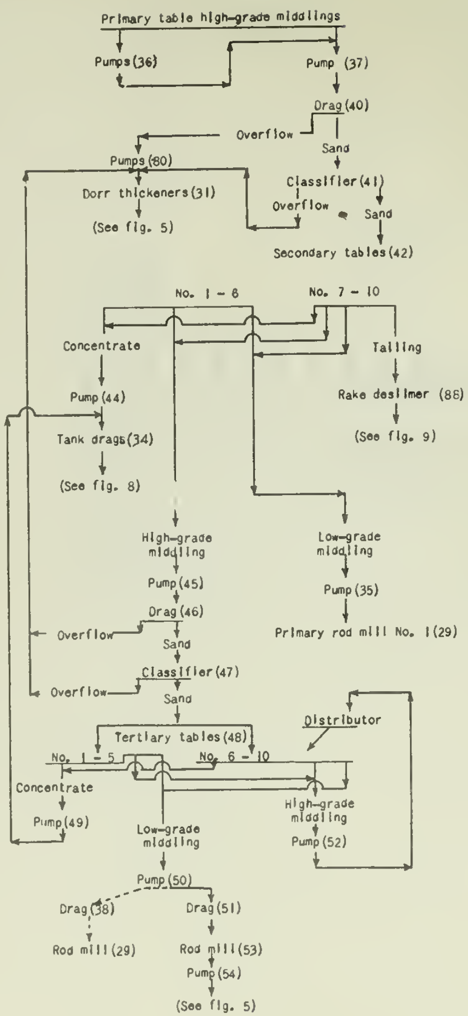
The 10 spigot products of each primary classifier are treated on 8 tables. The amount of feed ranges from 50 to 75 tons per table per 24 hours. The table feeds are about 25 per cent solids. From 4 to 15 gallons of wash water per minute are used on each table. The mechanical and operating details of the concentrating tables are given in Table 13.

Linoleum and quick-setting cement were used formerly for deck-surfacing material, but both have been replaced by live-rubber covers. Rubber as thick as 1/4 inch has been tried but a thinner cover, 1/8 or 3/16 inch thick, is now used. At first rubber cement and tacks were used to hold the rubber cover in place, but now the practice is to tack the cover at one end, stretch it well by a weight applied to the other end, and finally tack the rest of the cover in place.

Table 13.-Data on concentrating tables

Flow-sheet serial	(32) (42) and (48)			
Table, make	Wilfley	Butchart	Deister-Plato	St. Joe
In use, each	76	19	2	19
Strokes per minute	270	255	270	240
In use, total	116			
Stroke length ... inches	7/8 to 1-3/8			
Drive, type	Belt-and-pulley			
Power, total ... horsepower	192			
Connected per table ... do.	1.65			
Deck:				
Size ... feet	5 by 15			
Slope, longitudinal ... degrees	0 to 1/2			
Slope, lateral ... do.	3 to 5-1/2			
Surfacing material	1/8 to 3/16 inch rubber			
Riffles, type	Straight, parallel to motion			
Material	Rubber, or wood and rubber			
Height ... inches	1/16 to 3/8			
Spacing ... do.	2			
Feed:				
Character	Spigot discharge			
Rate per 24 hours ... tons	50 to 75			
Pulp density ... per cent	25			
Wash water per minute ... gallons	4 to 15			

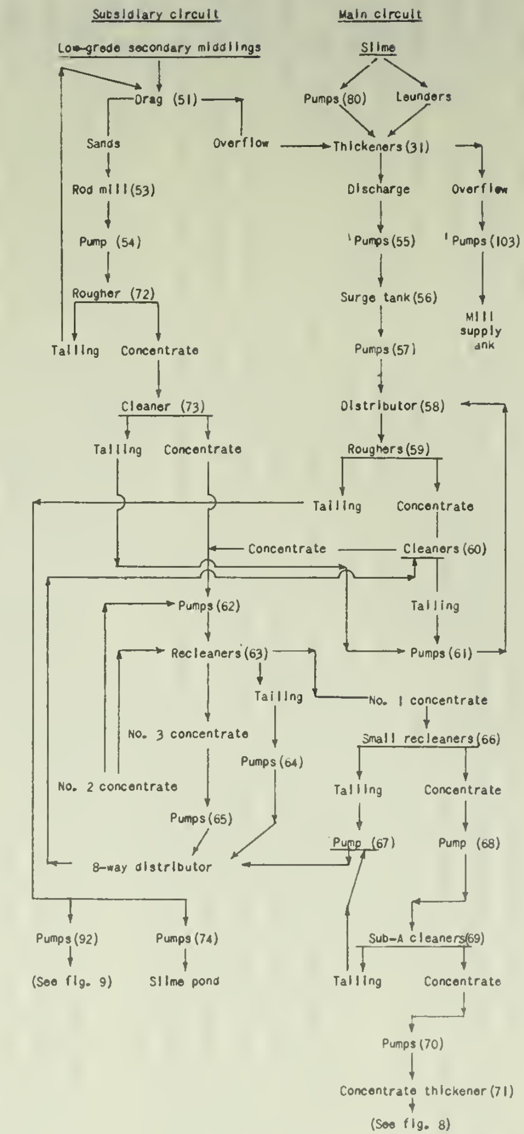
Rubber has proved more resistant to wear than linoleum and does not become uneven, as was the case with the cement deck. The porosity of the cement gradually allowed water to seep through to the boards underneath, causing them to warp and distort the surface. Cement is objectionable also because of its weight. The life of the rubber covers has not yet been determined but it is over two years. Some buckling occurs after long service but if properly laid the covers are quite satisfactory. One operator has determined that the life depends on the amount of absorbed water. After long service the tops contain from 2 to 14 per cent of moisture. The high moisture content is found in rubber with a low zinc oxide content. Tops containing 39 per cent of zinc oxide are good, while those with only 2 to 3 per cent are not so satisfactory.



Mechanical details

(29) 6 1/2 by 12 foot rod mill (see Table 9)
(31) Dorr thickeners (see Table 18)
(34) Tank drags (see Table 20)
(35) Low-grade middling pump of sec. 1 (see Table 23)
(36) Morris pumps (see Table 23)
(37) 4-inch Wilfley pump (see Table 23)
(38) Drag deslimer (see Table 20)
(40) Drag deslimer (see Table 20)
(41) Hydraulic classifier (see Table 11)
(42) Ten concentrating tables (see Table 13)
(44) 4-inch Morris pump (see Table 23)
(45) 4-inch Wilfley pump (see Table 23)
(46) Drag deslimer (see Table 20)
(47) Hydraulic classifier (see Table 11)
(48) Ten concentrating tables (see Table 13)
(49) 4-inch Morris pump (see Table 23)
(50) 4-inch Wilfley pumps (see Table 23)
(51) Drag deslimer (see Table 20)
(52) 4-inch Morris pump (see Table 23)
(53) 5 by 9 foot rod mill (see Table 9)
(54) 4-inch Wilfley pump (see Table 23)
(80) Three 8-inch Wilfley pumps (see Table 23)
(88) Reciprocating rake deslimer (see Table 26)

Figure 4.- Flow sheet of table middling section



Mechanical details

(31) Two 150-foot Dorr thickeners (see Table 18)
(51) Drag deslimer (see Table 20)
(53) 5 by 9 foot rod mill (see Table 9)
(54) 4-inch Wilfley pump (see Table 23)
(55) 5-inch Morris pumps (see Table 23)
(56) Cylindrical concrete tank, 24 feet in diameter by 8 feet deep
(57) 5-inch Morris pumps (see Table 23)
(58) 24-outlet distributor (see fig. 6)
(59) Twenty-three 36-foot St. Joe flotation machines (see Table 17)
(60) Twenty-three 12-foot St. Joe flotation machines (see Table 17)
(61) Four 5-inch Morris pumps (see Table 23)
(62) 4-inch Wilfley pumps (see Table 23)
(63) Two 42-foot St. Joe flotation machines (see Table 17)
(64) Two 4-inch Morris pumps (see Table 23)
(65) Two 4-inch Morris pumps (see Table 23)
(66) Two 12-foot St. Joe flotation machines (see Table 17)
(67) 4-inch Morris pump (see Table 23)
(68) 4-inch Wilfley pump (see Table 23)
(69) Two 2-cell Denver Sub-A flotation machines (see Table 17)
(70) 4-inch Morris pumps (see Table 23)
(71) 50-foot Dorr thickener (see Table 18)
(72) 36-foot St. Joe flotation machine (see Table 17)
(73) 12-foot St. Joe flotation machine (see Table 17)
(74) 5-inch Morris pumps (see Table 23)
(80) Three 8-inch Wilfley pumps (see Table 23)
(92) 5-inch Morris pumps (see Table 23)
(103) Water pumps (see Table 27)

Figure 5.- Flotation flow sheet

Straight riffles, laid parallel to the axis of motion, are used. They are of live rubber on the recent installations, but riffles of wood faced with rubber strips are found on the older tables. The riffles are held in place by copper tacks.

Secondary or Table Middling Circuit

The treatment of the high-grade primary table middlings is shown in the flow sheet, Figure 4. The middlings are gathered from all six primary sections and sent to a middling section with ten tables where they are classified and concentrated without preliminary grinding. A concentrate, high-grade middling, and low-grade middling are made on each table, and, in addition, a tailing is made on the last four.

The low-grade middling is reground in one of the primary rod mills and thus returns to the primary circuit. The high-grade middling is treated in a circuit of another group of 10 tables similar to the one described above for high-grade primary table middlings. In this circuit no tailing is made. It yields only concentrate, high-grade middling, and low-grade middling. The high-grade middling is circulated in the section by being distributed to the last five tables. The low-grade middling is dewatered in a drag deslimer and ground to flotation size in a 5 by 9 foot secondary rod mill. The sizing analyses of the feed and discharge of this mill are given in Table 14.

The rod-mill discharge is treated directly by flotation on a St. Joe rougher in the subsidiary flotation circuit, and the tailing is returned to the drag in closed circuit with the rod mill. This drag overflow, together with the overflow of the other two drags and hydraulic classifiers, is joined with the flotation feed that is pumped to the large Dorr thickeners. The drag sand returns to the rod mill. This part of the flow sheet is novel in that the rod-mill discharge goes to flotation and the flotation tailings return to the rod mill.

Table 14.-Sizing analyses of feed and discharge of secondary rod mill

Size, mesh	Weight, per cent	
	Feed	Discharge
Plus: 10	2.6	-
14	5.4	-
20	8.4	-
28	9.3	-
35	10.4	-
48	8.5	-
65	15.6	1.4
100	11.8	2.6
150	10.5	5.9
200	6.3	8.4
325	7.9	31.7
Minus: 325	3.3	50.0
Total......	100.0	100.0

FLOTATION

Flotation is in a main circuit and in a subsidiary circuit. The main circuit has 23 rougher-and-cleaner units and six recleaners; the subsidiary circuit has one rougher-and-cleaner unit. With the exception of the final recleaners, the machines are a modification of the Forrester or Welsch pneumatic type and are called St. Joe low-pressure flotation machines. A rougher-and-cleaner unit consists of a 36-foot rougher in tandem with a 12-foot cleaner, both of which receive air from one long, horizontal header pipe at the end of which is a centrifugal blower; the unit is about 55 feet long. Each of two 42-foot recleaners has its own blower, but only one blower is required for two small 12-foot recleaners set in close parallel. The final recleaners are two 2-cell, 24-inch Denver Sub-A mechanical-type machines. All the pneumatic flotation machines are set parallel on one floor with the blowers in line at one end.

The subsidiary circuit receives reground low-grade table middlings from the gravity plant, and its products join the corresponding products in the main circuit. The main circuit feed is primary and secondary slimes, thickened in the two 150-foot Dorr thickeners. Rougher concentrates are cleaned four times to make final concentrates. The flotation flow sheet is given in Figure 5.

The primary feed is a pulp of about 20 per cent solids by weight, and it is distributed to the roughers by a 23-outlet distributor, shown in Figure 6. The cleaners are fed a pulp of lower density. Sizing analyses of feed, concentrate, and tailing are given in Table 15.

Flotation reagents are potassium xanthate, sodium cyanide, and cresylic acid or pine oil, or a mixture of the two. The device for feeding pine oil is sketched in Figure 7. Other reagents are fed in solution by pulley and scraper feeders. The flotation circuit is slightly alkaline, having a pH value of about 8.3.

Table 15.-Sizing analyses of flotation feed, tailing, and concentrate

Size, mesh	Weight, per cent		
	Feed	Tailing	Concentrate
Plus: 65	0.8	1.0	-
100	3.4	3.1	0.3
150	9.6	9.4	1.0
200	10.2	13.5	2.5
325	21.8	22.7	15.5
Minus: 325	54.2	50.4	80.7
Total....	100.0	100.0	100.0

In the main circuit xanthate and cresylic acid or pine oil are fed to the intake pipe of the pump that elevates the feed from the surge tanks to the distributor. Sodium cyanide is added to the sump of the pumps that elevate the cleaner concentrates to the first recleaners. Thus the conditioning periods are very short.

In the subsidiary circuit, xanthate and cyanide are added to the rod mill, and cresylic acid or pine oil to the sump of the pump that elevates the mill discharge to the flotation machine.

The amount of reagents per ton of flotation feed is given in Table 16.

Mechanical details of the St. Joe pneumatic and the Denver Sub-A mechanical flotation machines are given in Table 17. Data on the Dorr thickeners for dewatering flotation feed and flotation concentrates are in Table 18.

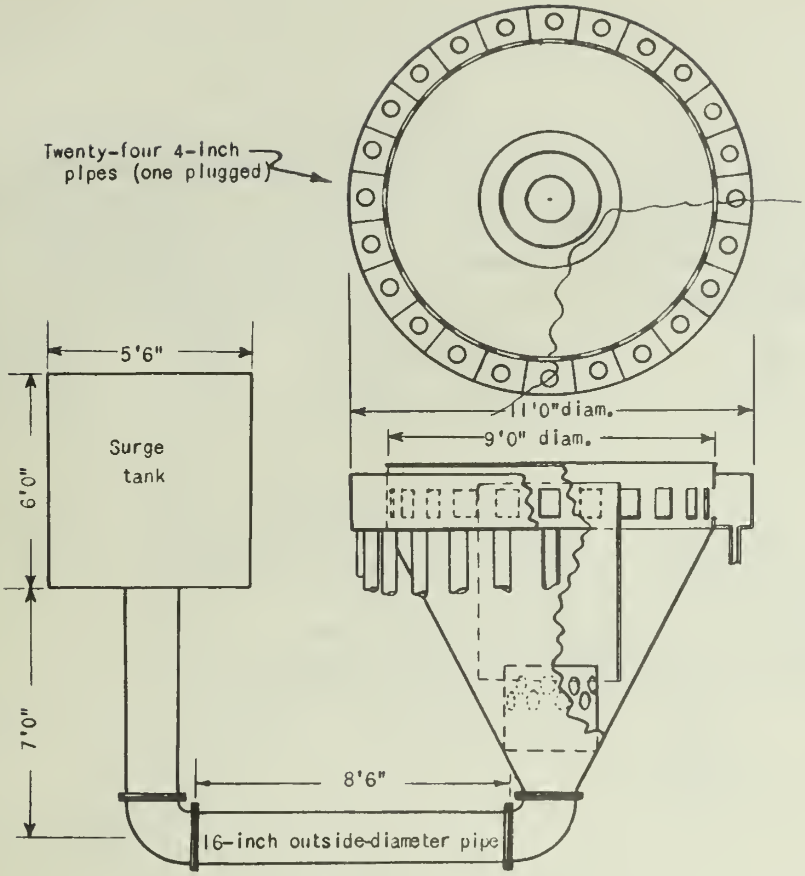


Figure 6.- Flotation feed distributor (58)

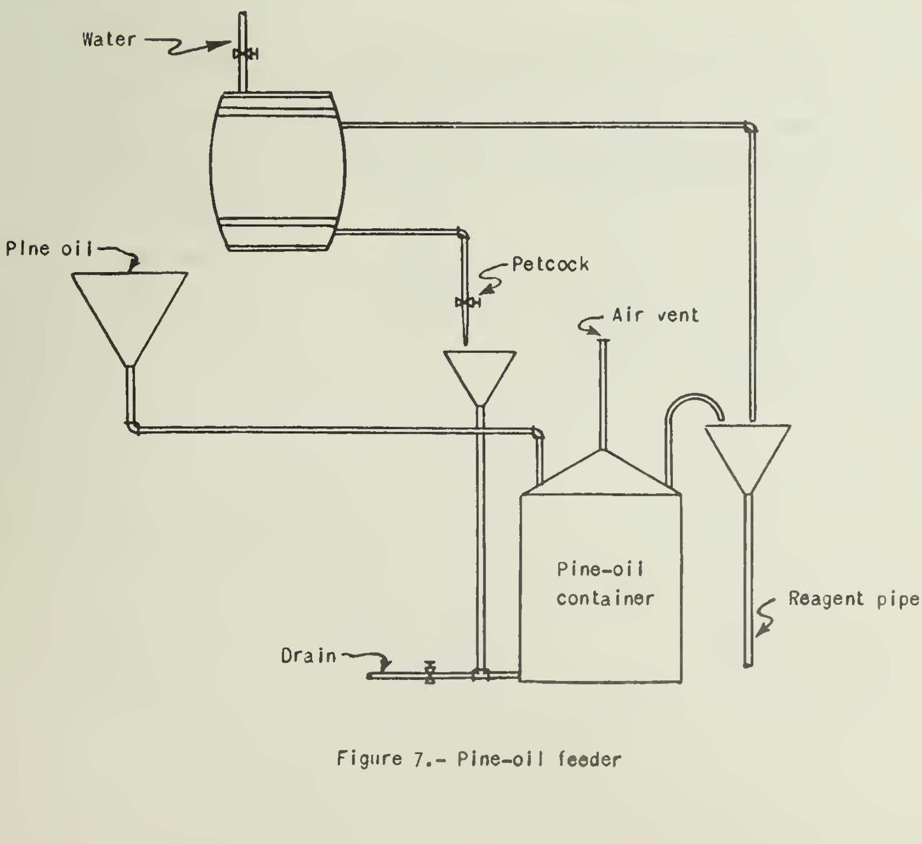


Figure 7.- Pine-oil feeder

Table 16.–Amount of flotation reagents per ton of flotation feed

Using cresylic acid		Using pine oil	
Reagent	Per ton of flotation feed, pounds	Reagent	Per ton of flotation feed, pounds
Cresylic acid	0.20	Pine oil	0.06
Potassium xanthate	.083	Potassium xanthate	.083
Sodium cyanide	.04	Sodium cyanide	.04

Table 17.–Flotation machine data

Flow-sheet serial	(59) (72)	(60) (73)	(63)	(66)	(69)
Machines, in use	24	24	2	2	Two, 2-cell
Type	Low-pressure pneumatic				Mechanical
Size	37 inches wide by 38-1/2 inches deep				44 by 59 inches
Manufacture	St. Joe	St. Joe	St. Joe	St. Joe	Denver Sub-A
Service	Rougher	Cleaner	Recleaner	Small recleaner	Final recleaner
Length ... feet	36	12	42	12	
Impellers, diameter ... inches	–	–	–	–	24
Overflow height:					
Tailing ... do.	18	18	21	19	–
Froth ... do.	27	24	32	39	21
Air header, diameter ... do.	16	16	16	16	–
Air pipes:					
Size ... inches	Diameter, 3; opening, 1/4 by 3-1/2				–
Number	48	12	60	18	–
Length	To within 5 inches of bottom of trough				–
Blower type	G. E. type F S 353, 3,500 r.p.m.				–
Air, per minute ... cubic feet	4,000 per unit		4,000 each	1,700	–
Pressure ... ounces	12		12	16	–
Motor capacity ... horsepower	18		18	12	Two, 5 each
Drive	Direct		Direct	Direct	4-belt texrope

Table 18.-Data on Dorr thickeners

Flow-sheet serial	(31)	(71) (75)
Thickeners, in use	2	2
Type	Traction	Central-shaft
Feed	Flotation feed	Flotation concentrate
Diameter ... feet	150	50
Depth ... inches	15	—
Speed ... r.p.m.	0.29	0.15
Solids:		
Per 24 hours, dry ... tons	1,454, each	—
In feed ... per cent	4.42	—
In discharge ... do.	48 to 55	—
In overflow ... do.	Trace	—
Area per ton per 24 hours ... square feet	11	—
Motor capacity ... horsepower	10	5
Drive, type	Direct-connected, gear reducer and spur gear	Motor, gear reducer, chain-and-sprocket

Screening and Mechanical Classification

Vibrating Screens

Screens are used at two places in the flow sheet. Dry screening is done in closed circuit with the roll grinding, and wet screening is carried on in closed circuit with the primary rod-mill grinding. Vibrating screens of the Leahy type are used throughout. Some of the screens have ball-bearing head motions but most of them are of the babbitted-bearing type. Woven wire with square openings is used for both wet and dry screening. The details of operation are given in Table 19.

Drag Deslimers

Desliming is by 21 drag deslimers and 2 tank drags. They are of the Esperanza chain type, and are driven by belt and spur gear. Some of them remove slime and divert it to the Dorr thickeners, and others serve only as dewaterers. No spray water is used, but water is added to the bowl of the drags treating the wet-screen undersize. The operating data on drags are given in Table 20. (The reciprocating rake drags used for tailing disposal will be treated in another section).

Table 20-a shows data from an efficiency test of a drag deslimer.

Mechanical Handling

The mechanical handling of the ore is by belt conveyors, bucket elevators, sand and slime pumps, and launders.

Table 19.-Screen data

Flow-sheet serial	(16)	(26)
Screens, in use	12	12
Width, effective ... inches	45	28
Length, effective ... do.	48	[1]55
Slope per foot ... do.	8-1/4	7-3/8
Tension application	Side	Side
Opening ... inches	0.17	0.12
Wire size, and diameter ... do.	No. 14, 0.08	No. 14, 0.08
Life ... days	[2]16	[2]16
Vibrator pulley speed ... r.p.m.	189 to 203	197 to 239
Vibrations:		
Per revolution	8	8
Per minute	1,512 to 1,624	1,576 to 1,912
Amplitude ... inches	1/8	1/8
Feed:		
Character	Roll discharge	Primary rod-mill discharge and section feed (wet)
Per hour per screen ... dry tons	107	35.4
Per square foot ... do.	7.1	3.3

1 - At each end 1½ inches and 3½ inches are doubled back; screen sheets are 60 by 34 inches.

2 - Working days of 24 hours each.

Belt Conveyors

The belt conveyors use 2,421 feet of belting over a total transporting length of about 1,200 feet. The connected power is 205 hp. The discharge of the primary crushers is transported by belts to the secondary storage bin. The discharge of this bin is conveyed from the feeders to the horizontal disk crushers, and the finished roll product is conveyed to and distributed over the concentrator storage bin. The deslimed table tailing is conveyed by belt to the pump box preparatory to mixing with part of the flotation tailings to be pumped to the tailing pond. The mechanical and operating details of the conveyors are given in Table 21. Specifications for conveyor belting call for 32-ounce duck, a friction of 25 to 30 pounds and a tensile strength of 3,500 to 4,000 pounds.

A belt conveyor transports the flotation concentrate from the dryer to the box-car loader, and two belts carry the gravity concentrates from the bin to the box-car loader; these are, however, small installations and are not described in Table 21.

Table 20.-Drag-deslimer data

Flow-sheet serial	(27)	(38)	(40) (46)	(51)	(34)
Drag deslimers, in use	12	6	2	1	2
Type	Esperanza	Esperanza	Esperanza	Esperanza	Tank drags[1]
Incline, length ... feet	15	14½	12	12	([2])
Slope per foot ... inches	5¼	8½	4¾	5½	7¾
Width ... do.	52	56½	55	68	22
Speed ... r.p.m.	30.7	33.9	37.0	33.0	31.8
Chain spacing ... inches	29	48	24	42	Single chain
Flights:					
Number	52	37	32	34	41
Spacing ... inches	8	12	12	12	16
Size ... do.	41x3x3/8	46x8x3/8	46x3x3/8	54x3x3/8	16x3x3/8
Feed:					
Character	Wet screen undersize	Low-grade primary table middlings	Middlings in retreatment circuits	Low-grade middlings from second middling section	Table concentrate
Per 24 hours ... dry tons	625	300	-	-	(Table 25)
Rake product ... do.	430	300	-	-	-
Overflow ... do.	195	-	-	-	-
Motor capacity ... horsepower	10, for 4 drags	5	5	5	5

1 - Cylindrical tank, diameter 9 feet, height 8½ feet.
2 - Flight length 57.6 feet.

Table 20-a.-Efficiency of a drag deslimer

Size, mesh	Weight, per cent		
	Feed	Discharge	Overflow
Plus: 6	0.4	0.7	-
8	6.5	9.4	-
10	9.4	14.4	-
14	7.1	11.0	-
20	4.3	6.6	-
28	6.0	8.9	-
35	7.2	8.7	0.1
48	8.0	11.0	.5
65	12.8	14.0	7.4
100	4.6	3.3	8.2
150	4.5	2.3	9.3
200	4.4	1.7	9.9
Minus: 200	24.8	8.0	64.6
Total	100.0	100.0	100.0

Table 20-a.-Efficiency of a drag deslimer (Continued)

Product	Tons per 24 hours: Feed	Tons per 24 hours: Discharge	Tons per 24 hours: Overflow	Removed in overflow, per cent
Solids	1,474	972	502	-
Water	[1]1,222	192	1,030	[2]84
Through 200 mesh	[3]380	[3]73	[3]307	[2]81
150 to 200 do.	66	16	50	[2]76
100 to 150 do.	68	21	46	[2]68
65 to 100 do.	70	31	39	56
48 to 65 do.	180	141	39	22

1 - Including fresh water.
2 - 78.5 per cent efficiency through 100 mesh.
3 - Correct numbers. Those not marked check very closely.
Note: Type of deslimer, Esperanza; Width, 46½ inches; slope, 8½ inches per foot. Flights, length 46½ inches; depth 3⅜ inches; thickness, ⅜ inch; spaced 12 inches center to center; number, 38. Speed, 44.8 feet per minute. Fresh water, 132 gallons per minute.

Table 21.-Belt-conveyor data

Product conveyed	Flow-sheet serial	Belt details: Length, feet	Belt details: Width, inches	Belt details: Ply	Belt details: Duck belting, ounces	Belt details: Rubber cover thickness, inches: Top	Belt details: Rubber cover thickness, inches: Bottom	Kind of drive	Slope per foot, inches	Speed of belt, f.p.m.	Motor horsepower	Average tonnage per hour
Discharge of three 9 E Telsmith primary breakers	(5)	95	36	5	32	3/16	1/32	Direct with reducer	Horizontal	394.3	5	329
Discharge of above conveyor	(7)	286	36	6	32	3/16	1/16	Belt and spur-gear	3-7/8	483.3	50	329
Feed to Symons horizontal disk crushers, west	(11)	178	24	5	32	1/4	1/16	do.	3-3/4	375.6	20	112
Feed to Symons horizontal disk crushers, east	(11)	178	24	5	32	1/4	1/16	do.	3-3/4	402.3	20	112
Undersize of dry screens	(17)	250	24	5	32	1/8	1/16	do.	Horizontal	320.0	20	224
Discharge of above incline to concentrator	(18)	450	24	5	28	3/32	3/32	do.	3-3/4	472.5	50	224
Mill feed, tripper conveyor	(21)	510	24	5	32	3/32	1/8	do	Horizontal	387.6	20	224
Table tails, east	(89)	124	22	(1)	32	3/32	1/8	Direct with reducer	1	265.4	5	42
Table tails, west	(89)	170	22	(1)	32	3/32	1/8	do.	1	272.0	5	42
Table tails, composite	(90)	180	22	(1)	32	3/32	1/8	Belt and spur-gear	1	374.4	10	84

1 - These belts are all of step construction, 3, 4, and 5 ply.

Bucket Elevators

Both dry and wet bucket elevators are used. Fourteen elevators require 1,585 feet of belting. The discharge of the Symons horizontal disk crusher is elevated to rolls, and the discharge of the rolls is elevated to the screens which are in closed circuit with the rolls. In the sampling plant the rejects of the cutters are returned to the mill feed by an elevator. The concentrator feed is united with the primary rod-mill discharge and elevated to the wet screens. In the flotation concentrate drying plant the discharge of the thickeners is raised to the filter by an elevator. Table 22 gives the mechanical and operating details of these installations.

The elevator belts are covered all around with cider cloth. A tensile strength of 3,500 to 4,000 and a friction of 20 to 24 pounds per square inch on the dry elevators and of 25 to 30 pounds on the wet elevators are specified. The life of both wet and dry elevator belts is three years or more.

Table 22.–Bucket-elevator data

Flow-sheet serial	(13)	(15)	(25)	(96)	(78)
Elevators, in use	2	4	6	1	1
Belting:					
Kind	Rubber	Rubber	Rubber	Rubber	Rubber
Duck, weight ... ounces	36	36	36	34	34
Ply	9	9	8	8	7
Cover thickness:					
Bucket side ... inches	3/16	3/16	1/8	1/8	1/16
Pulley side ... do.	1/8	1/8	1/8	1/8	1/8
Length ... feet	95	116	125	125	56
Width ... inches	28	28	20	16	16
Speed ... f.p.m.	344 to 410	415 to 450	340 to 352	269	460
Buckets:					
Number	67	81	83	62	37
Size ... inches	24 x 8 x 8 1/2	24 x 8 x 8 1/2	18 x 8 x 8 1/2	14 x 8 x 7 1/2	14 x 8 x 7 1/2
Spacing ... do.	17	17	18	24	18
How attached	Bolted with two rows of 3/8-inch bolts; strip of belting used to hold bottom of cups out			One row of bolts	
Pulley:					
Head, diameter and face ... inches	48, 30	48, 30	42, 22	36, 18	36, 18
Shaft, diameter ... do.	4 15/16	4 15/16	[1]4 15/16	3 7/16	3 7/16
Boot, diameter and face ... do.	30, 30	30, 30	30, 22	24, 18	24, 18
Elevator inclination	Vertical	Vertical	Vertical	Vertical	Vertical
Feed:					
Character	Symons discharge	Roll discharge	Wet screen feed	Mill feed, sample reject	Thickened flotation concentrates
Pulp per hour ... tons	112	322	111	3.5	8.5
Solids ... per cent	(2)	(2)	64.1	(2)	69.2
Motor capacity, horsepower	50	50	[3]20	[4]50	5
Drive	Belt and spur gear				Direct with speed reducer

1 - The bore of the pulley is 4 15/16 inches, but the bearings are 4 7/16.

2 - Run-of-mine ore contains 2.75 to 3.0 per cent of moisture.

3 - A 20-hp. motor on each wet elevator also drives two Leahy screens

4 - The 50-hp. motor also drives inclined conveyor from secondary plant to concentrator.

Table 23. - Operating details of sand slime pumps

Product	Primary table concentration				First middling treatment		Second middling treatment				Flotation										Tailing disposal	
	Low-grade middling	High-grade middling		Concentrate	High-grade middling	Concentrate	High-grade middling	Concentrate	Low-grade middling		Dorr thickener feed	Dorr thickeners discharge	Surge tank discharge	Cleaner tailing	Cleaner concentrate and recleaner No. 2 concentrate	Recleaner tailing No. 3	Recleaner concentrate No. 3	Small St. Joe and Sub-A tailing	Small St. Joe concentrate	Sub-A concentrate	Flotation tailing	Combined flotation and gravity concentrate.
Flow-sheet serial	[illegible]	(36)	(37)	(33)	(45)	(44)	[illegible]	(49)	[illegible]	(54)	[illegible]	[illegible]	[illegible]	(61)	(62)	(64)	(65)	(67)	[illegible]	(70)	(74)	(92) (93)
Pumping: Source	Basement	Basement	Basement	Basement	Basement	Basement	Basement	Basement	Basement	Rod-mill floor	Basement sump	Dorr thickeners	Surge tank	Rod-mill floor	Rod-mill floor	Rod-mill floor	Rod-mill floor	Rod-mill floor	Rod-mill floor	Rod-mill floor	Basement	Basement
Disposal	Dewatering drags	Pump (37)	1st middling section drag	Tank drags	2nd middling section, drag	Tank drags	Tables of slime section	Tank drags	Middling rod mill feed drag	Flotation machine	Two 150-foot Dorr thickeners	Surge tank	Distributor	Distributor	Recleaner	Cleaners distributor	Cleaners distributor	Cleaners distributor	Sub-A machines	Filtering end drying plant	Slime pond	Slime pond
Pumps, in use	12	5	1	6	1	1	1	1	1 in use 1 spare	1	3	2 in use, 1 spare	1 in use, 1 spare	4	2 in use, 2 spares	2	2	1	1	1 in use 1 spare	2 in use, 1 spare	2 in series
Manufacture	Wilfley	Morris	Wilfley	Morris	Wilfley	Morris	Morris	Morris	Wilfley	Wilfley	[illegible]	Morris	Morris	Morris	Morris	Morris	Morris	Morris	Wilfley	Morris	Morris	Morris
Size . . . . inches	4	(Four, 3 (One, 4	4	4	4	[illegible]	4	4	4	[illegible]	[illegible]	5	5	5	4	4	4	4	4	4	5	5
Motor . . . horsepower	20	10	35	10	20	10	10	10	20	10	100	40	40	[illegible]	10	10	10	[illegible]	10	20	40	75
Speed . . . r.p.m.	860	860	865	900 or 860	900	870	870	870	865	870	860	870	670	650	870	870	860	865	850	1130	870	1160
Intake piping: Head . . . . feet	2 1/2	1/2	1 1/2	1/2	2 1/2	1/2	1/2	[illegible]	3	[illegible]		19	4	3	3	3	3	3	3	3	8	3
Diameter . . . inches	6	Four, 4 One, 3	6	4	6	4	4	4	6	[illegible]	8	6 and 10	8	5	6 and 4	4	4	4	6	6	8	6
Length . . . feet	1/3 to 1	1/3 to 1	1	1/3 to 1/2	3/4	1/3	3/4	2/3	1	[illegible]	4 1/2	3 1/2 of 6-inch 87 of 10-inch	21	1 1/2	1/2 of 6-inch, 1/3 of 4-inch	1	1	1	2/3	1/2	38	1/2
Fittings	None	None	None	None	None	None	None	Valve	Valve	None	Valve and 45°	Valves, Y, and increasers	90°, and valves	Valve	Reducer, and valve	Valve	Valve	Valve	Valve	Valve, and reducer	90°, tees, and valve	None
Discharge piping: Vertical lift . . feet	20	4 1/2 to 7 1/2	23	28	33 1/2	[illegible]	20 1/2	28	23 1/2	23	41	27	43	41	35	32	32	32	17	7	---	---
Diameter ....inches	Some 4, most 6	Two 3, three 4	4	Three 3, three 4	4, and 6	4	4	4	4	4	8,10,12	---	---	---	---	---	---	---	4	4	One 12, one 8	8
Length . . . . feet	90 to 157	61 to 156	93	53 to 146	107 of 4-inch, 50 of 6-inch	158	76	137	135	80	6 of 8-inch 16 of 10-inch, 406 of 12-inch	210	94	73 to 151	96 to 81	108 to 122	118 to 122	122 1/3	42	513	3,500	2,700
Fittings	90°, 45°, and hose	90°, 45°, and hose	90°, 45° and hose	90°, 45°, tees, and hose	90°, 45°, and hose	90°, 45° and hose	90°, tees and hose	90°, tees, and 45°	90°, 45°, and hose	Valve, and 90°	Increasers 90°, 45°	90°, discharge 1 1/2-inch diam.	90°	45°, 90°, and tees	45°, 90°, and hose	45°, 90°, and hose	Hose, 45°, 90°, and valve	45°, 90° and hose	90°, and hose	90°, 45°, and hose	90°, and tees	90°, and tees
Feed: Rate per 24 hours . tons	225	50	300	17	150	10	10	6	122	122	667	1,600	3,200	600	----	----	----	----	----	----	----	----
Solids .. per cent	12	24.7	24.7	5.3	24.7	5.3	24.7	5.3	24.7	20 -25	4.5	45-50	20	10	12	----	----	----	----	----	22	35-38

Malleable iron buckets of the "AA" type, having a reinforced front edge and corners, are used. The life of the wet elevator buckets is about 11 months. Formerly the dry elevator buckets wore away at the front corners, but now the lip is reinforced by welding on an edge of Hascrome or Stoody rod.

Solid boot pulleys are used instead of the rimless pulleys employed in some of the other mills of the district. The operation of the solid pulley has been satisfactory, and the wear not excessive. Both wooden and oil-packed steel bearings are used in the boots.

Sand and Slime Pumps

Pumps are used for the dispostion of sands and slimes. (The water pumps will be discussed elsewhere). They are all of the centrifugal type, 3, 4, and 5 inch Morris pumps, and 4 and 8 inch Wilfleys, and are direct driven by induction motors. No water is used on the Wilfleys, but clear water is used on the stuffing boxes of the Morris pumps. The details of sand and slime pump operation are given in Table 23.

The two makes of pumps are used more or less interchangeably. However, certain features in the designs have been recognized as more adaptive to one service than another. The positive suction at the intake of the Morris pump enables it to lift fine flotation products from a lower level. On the other hand, the gravity intake of the Wilfley has led to its use on coarser and heavier products that can not be lifted by suction. The absence of suction by the Wilfley pumps prevents the air locking that causes surging in other pumps. It is claimed that the cost of upkeep of the Wilfley pumps is considerably less, and there is a tendency to use them for coarser, heavier, or more abrasive solids.

Launders

Launders of the box-conduit, hose, and pipe type are used. The box-conduit launders are used for large tonnages and for smaller tonnages if more convenient than the other types. The principal uses of this type of launder are for transporting the desliming-drag discharges to the 2-way revolving distributors feeding the hydraulic classifiers, and for carrying the spigot discharges of the classifiers to the concentrating tables. These launders are lined with cast iron or rubber; a flat piece is laid in the bottom and a triangular piece of wood moulding is used to line the corners. Rubber hoses are used for uniting small tonnages in a common launder or pump sump. The pipe type is used only for slime. The data on the box-conduit launders are given in Table 24. Other launders are used for the disposition of the table products; they are not included.

Table 24.-Box-conduit launders to hydraulic classifiers and concentrating tables

Product	Desliming drag, rake discharge	Flow-sheet serial	Hydraulic-classifier spigot products								Flow-sheet serial
Transportation:											
From	Drag deslimer	(27)	Hydraulic classifier								(30)
To	2-way distributor	(28)	8 primary tables of each half section								(32)
Launders, in use	6		96								
Width ... inches	12		4 3/4								
Depth ... do.	9		3								
			Table number								
			1	2	3	4	5	6	7	8	
Slope per foot, minimum ... do.	2		3 15/16	6 1/2	7	4 3/8	2 1/4	2 1/2	2 1/8	2 1/4	
Feed:											
Solids ... per cent	22.4		34.8	34.0	33.6	20.0	37.3	38.0	39.3	28.8	
Rate per 24 hours ... tons	850		(See Table 10)								

DEWATERING THE CONCENTRATES

Table and flotation concentrates are dewatered separately. Figure 8 is the flow sheet of the dewatering of these products.

Gravity Concentrates

The table concentrates are first dewatered in two tanks equipped with drag mechanisms. The tanks give a longer periphery for overflow and more depth than is customary on other drags. The water overflows and joins the flotation concentrates, the dewatering of which is discussed later. The drag discharge is further dewatered in a Dorrco filter. The filtered concentrates drop into two bins from which they are conveyed to a Stephens-Adamson box-car loader and loaded for shipment. The details of the operation of the tank drags are given in Tables 20 and 25.

Table 25.-Operation of tank drags for table concentrates

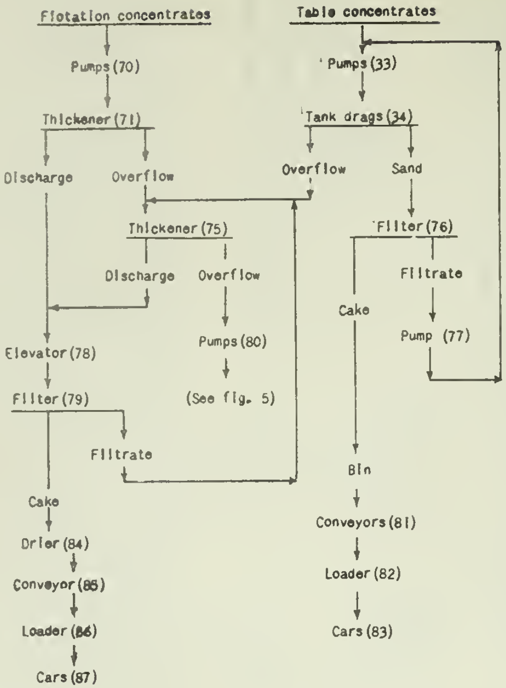
(See also Table 20)

Flow-sheet serial	(34)
Feed:	
Rate per 24 hours, wet.....tons	2,796
Rate per 24 hours, dry.....do.	148
Water in.....do.	2,648
Solids.....per cent	5.3
Rake discharge (filter feed):	
Rate per 24 hours.....tons	165
Water in.....do.	17
Water rejected.....do.	2,631
Solids.....per cent	90
Overflow:	
Rate per minute.....gallons	439
Solids per gallon.....grams	2
Solids per 24 hours.....tons	1.4
Assay, lead in solids.....per cent	60.7

The Dorrco filter feed of about 145 tons per 24 hours contains 10.2 per cent of moisture and is reduced to 134 tons with 2.8 per cent moisture; this product is the finished gravity concentrate. The water rejected, 11 tons per 24 hours, is 74.6 per cent of the water in the filter feed; 25.4 per cent of the water, or 3.75 tons is still contained in the finished product.

The Dorrco filter is 10 feet in diameter by 2 feet long and is operated at 1 r.p.m. The filter cloth consists of burlap supporting Victor twill, which is discussed later in the operation of the Oliver filter. The cloth is held in place by ropes. The filter is driven by a 5-hp. induction motor, 1,140 r.p.m., through a Philadelphia Gear Works speed reducer, ratio 12.75:1, connected with the filter by a chain-and-sprocket drive.

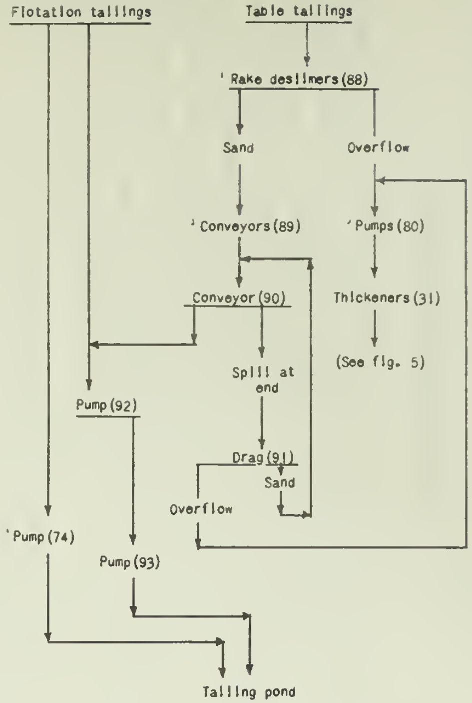
The vacuum pump used in connection with the filter is an Ingersoll-Rand with a 14 by 5 inch vacuum cylinder. It is operated by a 15-hp. induction motor with Texrope drive. From 12 to 16 inches of vacuum is maintained.



Mechanical details

(33) 4-inch Morris pumps (see Table 23)
(34) Two tank drags (see Tables 20 and 25)
(70) 4-inch Morris pumps (see Table 23)
(71) 50-foot Dorr thickener (see Table 18)
(75) 50-foot Dorr thickener (see Table 18)
(76) Dorrco filter, 10-foot diameter by 2 feet long, 1 r.p.m., 5-hp. motor
(77) 2-inch Gould pump, 1,720 r.p.m., 5-hp. motor
(78) 16-inch bucket elevator (see Table 22)
(79) Oliver filter, 11 feet 6 inches in diameter by 12 feet long, 0.12 r.p.m., 5-hp. motor
(80) Three 8-inch Wilfley pumps (see Table 23)
(81) Two 24-inch belt conveyors, 9 feet long; 3-hp. motor; Reeves No. 0 for both
(82) Stephens-Adamson box-car loader
(83) Box cars, 40 to 50 tons capacity
(84) Lowden drier, 12 by 28 feet long, 11-inch stroke, 2.4 s.p.m., 5-hp. motor; gas-fired
(85) 24-inch belt conveyor, 24 feet long; 40 feet per minute chain drive at tail pulley from drier
(86) Stephens-Adamson box-car loader
(87) Box cars, 40 to 50 tons capacity

Figure 8.- Flow sheet of concentrate dewatering



Mechanical details

(31) Two 150-foot Dorr thickeners (see Table 18)
(74) 5-inch Morris pumps (see Table 23)
(80) Three 8-inch Wilfley pumps (see Table 23)
(88) Four rake classifiers (see Table 26)
(89) Two 22-inch belt conveyors (see Table 21)
(90) One 22-inch belt conveyor (see Table 21)
(91) 23-inch single-chain drag desliner; 16 feet long, slope 5.9 inches; 48 flights, 19 by 2 by 1/4 inch; spacing 8 inches; 16.2 feet per minute; 3-hp. motor
(92) 5-inch Morris pump (see Table 23)
(93) 5-inch Morris pump (see Table 23)

Figure 9.- Flow sheet of tailing disposal

A Connersville blower, size 20 B, supplies the pressure for removing the cake. It is operated by a 3-hp. motor, 1,140 r.p.m., direct-connected. A pressure of 2 to 4 pounds, depending on the condition of the filter cloth, is maintained.

A small Goulds pump, direct-connected to a 5-hp. motor, running at 1,720 r.p.m., removes the filtrate to a pump that feeds the tank drag.

Flotation Concentrates

The flotation concentrates, together with the overflow of the tank drags, are dewatered in a 50-foot Dorr thickener and the overflow is settled in another thickener of the same size. The overflow of the second concentrate thickener is returned to the pumps that elevate the slime to the big flotation-feed Dorr thickeners. The thickened concentrates from both concentrate thickeners are elevated to an Oliver filter. The filtrate is returned to the second concentrate thickener, and the cake is dried on a Lowden dryer. The dried concentrate is conveyed to the box cars and loaded by a Stephens-Adamson box-car loader.

Two 8-hour shifts with four hours intervening are necessary for dewatering and drying the flotation concentrates. The operating time is adequate to handle the 3-shift production of the flotation plant.

Flotation produces 3.9 dry tons of concentrate per hour. This is pumped to the first 50-foot Dorr thickener in a pulp of 13.3 per cent of solids, so that the gross weight is 29.3 tons per hour. Thus, 25.4 tons of water per hour goes to the first Dorr thickener.

The time of operation of the filter is 16 hours per day, and the amount treated by the drying plant is 5.9 dry tons per hour of operation.

The concentrate thickeners discharge 8.5 tons of pulp of 69 per cent solids per hour for a period of 16 hours, and for the remaining two 4-hour periods nothing is drawn from the tank. Hence, of the 25.4 tons of water per hour sent to the Dorr thickener, 359 tons for 16 hours and 210 tons for 8 hours, or a total of 569 tons for the 24-hour shift, is rejected; 41 tons remain with the feed of the Oliver filter.

The Oliver filter feed of 8.5 tons per hour in a pulp of 69 per cent solids is reduced to 6.7 tons per hour in a product with 12.1 per cent of moisture. The water rejected in the filtering is 1.8 tons per hour.

The Lowden dryer reduces the moisture of the cake from 12.1 to 7.3 per cent. The dried concentrate is 6.4 tons per hour, and the water rejected by the dryer is 0.3 tons per hour. The amount of water rejected in the successive steps is as follows:

Source	Tons per 24 hours	Weight, per cent
In Dorr tank	569	93.3
In Oliver filter	28	4.6
In Lowden dryer	5	.8
Remaining in concentrate	8	1.3
	610	100.0

The Oliver filter is 11 feet 6 inches in diameter and 12 feet long, and is driven through a worm gear at a speed of 0.12 r.p.m. Two covers are used; they consist of a burlap cover and a 14 by 37.5 foot Victor twill, No. 223. The wire for holding the covers is a No. 12 hard-drawn, galvanized steel wire would spirally with a pitch of 3/4 inch. The life of the filter cloth is 11 months. The long life is probably due to its being given every 7 or 8 weeks an acid wash in which 114 pounds of commercial hydrochloric acid is used. The bowl of the filter is of the shallow semi-cylindrical type. To keep the concentrates in suspension in the bowl a reciprocating paddle is used.

The two vacuum pumps, one of which is a spare, are the Oliver make, Type M 0, 14 by 8 inch cylinder, 300 r.p.m. Each has 400 cubic feet displacement and is operated by a 25-hp. motor. The vacuum is maintained at 15 to 16 inches.

From 1 to 3 pounds air pressure is used to displace the cake. The pressure depends on the condition of the canvas; very little is required for a new canvas. The cake is drier with an old canvas than when a new one is used; with a new canvas the moisture in the cake may run as high as 20 per cent, but the cake is generally thicker.

The Lowden dryer is 12 feet wide and 28 feet long and is driven by a 5-hp. motor. The rake mechanism makes a forward stroke of 11.5 inches at the rate of 2.4 r.p.m. and has a lift of 3 inches. Natural gas is used for heating.

A flat arch over the fire box of the Lowden dryer is employed. Formerly the ordinary arch construction was used and considerable trouble was encountered from failure. The flat arch has eliminated this fault. The method of construction is as follows:

There are three horizontal I-beams over the fire box, perpendicular to the long axis of the dryer. Six smaller I-beams are bolted to the underside of these and perpendicular to them. On the smaller I-beams are hung six rows of 24 fire bricks. Each brick is 10.5 inches wide, 5 inches long, and 12 inches high and is made with a center groove to fit the I-beams so that the top of the brick is flush with the upper horizontal part of the beam. All of the bricks are fitted closely together.

DEWATERING AND DISPOSAL OF TAILINGS

Only the table tailings are dewatered prior to disposal. They are then mixed with a portion of the flotation tailings and pumped to the slime ponds. The rest of the flotation tailings are pumped separately to waste. The flow sheet of the dewatering and disposal of tailings is given in Figure 9. The table tailings are dewatered in four Dorr rake deslimers. The details of the rake deslimer operation are given in Tables 26 and 26-a.

Since the adoption of the method of pumping the combined tailings to waste, the classifiers serve principally as deslimers, whereas with the old system of conveying the tailings to waste, their principal use was for dewatering. The rake overflow is joined with the flotation feed and the sands are collected, repulped, and pumped to waste. A second pump in series is used to keep up the velocity and prevent settling of the sands in the pump line leading to the tailing pond.

Table 26.-Reciprocating rake-deslimer data

Flow-sheet serial	(88)
Classifier:	
Type	Duplex reciprocating rake
Size ... feet	8 1/3 by 30
Slope per foot ... inches	4
Lift ... do.	4
Stroke ... do.	11
Speed per minute ... strokes	17.6
Motor ... horsepower	10
Feed:	
Rate per 24 hours, wet ... tons	1,700
Rate per 24 hours, dry ... do.	381
Solids ... per cent	22.4
Rake discharge:	
Rate per 24 hours, wet ... tons	498
Rate per 24 hours, dry ... do.	374
Solids ... per cent	75.2
Overflow:	
Rate per 24 hours, wet ... tons	1,202
Rate per 24 hours, dry ... do.	7
Solids ... per cent	0.57
Water removed ... do.	90.6

Table 26-a.-Sizing analyses of rake-deslimer products

	Weight, per cent		
Size, mesh	Feed	Rake discharge	Overflow
Plus: 28	5.1	5.2	-
35	13.1	13.0	-
48	25.0	27.1	0.6
65	35.9	39.2	2.3
100	10.2	8.2	9.6
150	6.5	4.6	17.2
200	2.6	1.9	18.3
Minus: 200	1.6	.8	52.0
Total	100.0	100.0	100.0

The tailing-pond dam consists of current table tailings, and it is raised from time to time to keep pace with the filling of the pond. To build the dam, or to raise it, clear water, in place of flotation tailings, is pulped with the table tailings; the sand deposits more rapidly on the top of the embankment when pulped in this manner. The flotation tailings are deposited simultaneously on the inner side to seal the bank. The dam requires attention about once a year.

The pond overflow is recovered by a vertical concrete tower, 5.5 feet square, located within the impounded area. The lower portion is concrete on four sides and the upper part

has only two opposite sides extended. These sides are built with notches at both ends, and timbers are filled in to raise the weir as often as necessary to obtain a clean overflow. The tower is bottomed by a horizontal tunnel extending through the dam. Through this tunnel the intake lines of the pumps for returning the clear water to the mill are laid; pipes are connected with the bottom of the vertical tower which forms a sump (see fig. 11).

SAMPLING

The mill feed sample is taken as the ore drops to the tripper conveyor over the mill bin. The further treatment is shown in the flow sheet of the sampling plant, Figure 10. over a period of five months the assay determined by sampling the mill feed checked the lead as determined by assay and weight of concentrates and tailings with an error of only one in 200; that is, with an error of 0.02 per cent of lead when the mill feed was 4 per cent of lead.

Two shift samples, table tailings and flotation tailings, are taken by mechanically operated cutters. Fifteen other shift samples -- new flotation feed, rougher feed, cleaner concentrate, cleaner tailing, recleaner concentrate No. 1, recleaner concentrate No. 2, recleaner concentrate No. 3 and recleaner tailing, finished concentrate (Denver Sub-A concentrate), Denver Sub-A tailing, reground table middlings (subsidiary flotation circuit feed), subsidiary flotation circuit tailing, table concentrates, A-middling-section concentrate, and B-middling-section concentrate -- are sampled by hand. For the hand samples at least six cuts per shift are taken. The mechanical samplers cut the flotation and table tailings every 18 minutes.

Concentrates are sampled in the cars and also in the mill. In the loading, the table concentrates are leveled in the cars to a depth of 22 inches and flotation concentrates to a depth of 30 inches; the space in front of the car doors is left clear. Three rows of four vertical holes are sampled on either side of the door. The samples are taken by a conical cutter 1 inch in diameter at the bottom and 2 inches in diameter at the top. The entire amount of concentrate held in the cutter is taken as the sample and is cut down in the sample room. Two separate rooms are used for preparing mill samples for assay. The "concentrate room" is used for table and flotation concentrates, and the "ore room" for ore, middlings, and tailings.

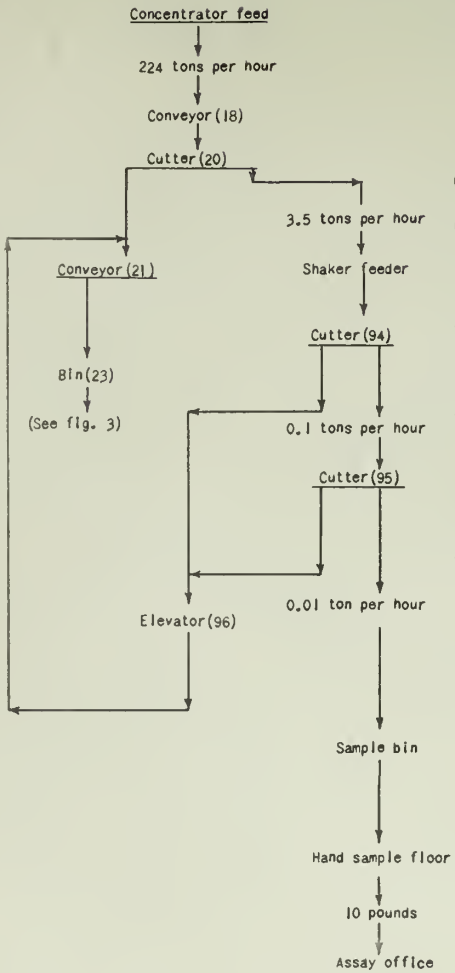
MILL WATER

The mill water is obtained from four sources, the mine, the large Dorr thickeners, the tailing pond, and the table low-grade middling drags. The circulation of the water is shown in Figure 11.

For 11 hours each day the make-up water is pumped from the mine at the rate of 2,000 g.p.m. The total water in circulation is about 10,000 g.p.m., which on the basis of 5,000 tons of ore per 24 hours is a water-to-ore ratio of 12 to 1 by weight.

The thickener overflow is piped to a sump and is pumped to the mill supply tank. The mine water, when pumped, is delivered to the same sump and the overflow is diverted to the tailing pond. The tailing-pond water is returned to the mill when the supply of mine water is insufficient or when the mine pump is not in operation.

The overflow of the drags that dewater low-grade primary table middlings is reused in the mill to dilute the wet elevator feed, to further dilute the same product as it is fed to the drag deslimers, and to dilute the rake discharge of the drags so that it can be laundered to the hydraulic classifiers. The operating data of the pumps for circulating mill water are given in Table 27.



Mechanical details

(18) Belt conveyor (see Table 21)

(20) Vezin-type sample cutter, 1-wing, 4-inch opening, 14.5 r.p.m.

(21) Belt conveyor (see Table 21)

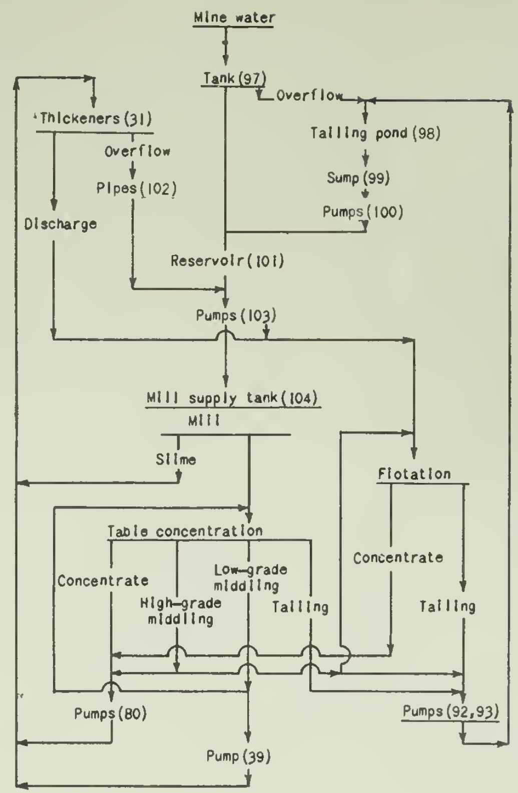
(23) 8,000-ton concentrator bin

(94) Vezin-type cutter, 2-wing, 3-inch opening, 15.6 r.p.m.

(95) Vezin-type cutter, 4-wing, 3-inch opening, 13.6 r.p.m.

(96) 16-inch bucket elevator (see Table 22)

Figure 10.– Flow sheet of sampling plant



Mechanical details

(31) Two 150-foot Dorr thickeners (see Table 18)

(39) 8-inch Wilfley pump (see Table 27)

(80) Three 8-inch Wilfley pumps (see Table 23)

(92) Pump (see Table 23)

(93) Pump (see Table 23)

(97) 500,000-gallon tank, 50 feet in diameter by 33 feet high

(98) Tailings pond

(99) Overflow tower and sump, 5 feet square

(100) Two 10-inch Allis-Chalmers pumps (see Table 27)

(101) Reservoir

(102) 18-inch tile pipes from thickeners, 27-inch pipe to sump

(103) Two 16-inch Alberger pumps (see Table 27)

(104) 108,000-gallon tank, 24 by 24 by 12 feet

Figure 11.– Flow sheet of water circulation

Table 27.–Pumps for mill water

Flow-sheet serial	(103)	(103)	(103)	(100)	(39)	(103)
Pumps, in use	1	1	1	2 (1 spare)	1	2
Manufacture	Dayton-Dowd	Dayton-Dowd	Jeanesville	Allis-Chalmers	Wilfley	Alberger turbine
Size, inches	16	6	12	10 x 10	8	16
Capacity per minute, gallons	8,000	8,000	–	2,000	–	5,000, each
Serving	Mill supply tank	Flotation and stuffing boxes on 5-inch pumps	Spare to mill tank	Slime pond to reservoir	–	Spares to mill tank
Motor:						
Type	Synchronous	Induction	Synchronous	Induction	Synchronous	Synchronous
Capacity, horsepower	250	75	150	50	100	250
Speed, r.p.m.	1,200	1,755	600	1,740	900	600
Intake piping:						
Head, feet	4	4	4	–	–	–
Diameter, inches	20	8	12	–	–	–
Length, feet	4	4	1	–	–	–
Fittings	90°, and valve	90°, and valve	90°, and valve	–	–	–
Discharge piping:						
Vertical lift, feet	76, from water surface to discharge	–	76	–	–	76
Diameter, inches	20	6	12	–	–	–
Length, feet	76	76	76	–	–	76
Fittings	90°, and valves	90°, and valves	90°, and valves	90°, and valves	90°, valves, and tees	90°, and valves

POWER

The electric power is 3-phase, 60-cycle current transmitted to the mill at a potential of 7,200 volts. It is stepped down to 440 volts for all motors and to 110 for lighting circuits. With the exception of five synchronous motors used on mill water the motors are all of the induction type. A 125-volt d.c. generator, 400 amperes, 50-kilowatt compound wound, driven by a 75-hp. induction motor at 1,150 r.p.m., supplies the direct current for excitation of the synchronous motors and for the two electromagnets in the primary-crushing plant.

Mechanical transmission of power is by many methods. Belt-and-pulley drives, either direct or by means of a line shaft, are used on the primary breakers, secondary crushers, rolls, screens, and concentrating tables. Belt drives with Reeves speed reducers are used on some feeders. Belt and spur gears are used on belt conveyors, drag deslimers, and bucket elevators. The rod mills are driven by herringbone gear and pinion direct-connected by a flexible coupling to the motors. Direct-connected motors are used on the flotation blowers. Direct-connected gear reduction units are used at various places on feeders, thickeners, etc. Bevel gears are used on some conveyor belts in connection with the concentrator feeders. Chain-and-sprocket drive with gear reducers is used on the 50-foot Dorr thickener.

RECOVERY AND LEAD LOSSES

The recovery of lead is between 96 and 97 per cent. The average assays of the feed, tailing, and concentrate for 1926 to 1930, respectively, and for the first five months of 1931 are given in Table 28. During the 6-year period the loss in the tailings was reduced to about one-third of what it was originally. This emphasizes the benefits of research, one of the results of which was the introduction of classification.

A complete analysis of a table concentrate and a flotation concentrate is given in Table 29. The representative lead assays of the shift samples of other products in the mill are shown in Table 30. The metallurgical data of the overall operation are given in Table 31. The screen analyses of table and flotation tailings are given in Table 32.

Table 28.–Average assay, lead in mill feed, tailings, and concentrates, per cent

Product	Assay, lead, percent					
	1926	1927	1928	1929	1930	1931[1]
Feed	3.91	3.92	3.72	3.51	3.55	3.43
Tailing:						
Table	.40	.23	.16	.12	.11	.11
Flotation	.19	.20	.16	.16	.15	.12
Combined	.33	.22	.16	.15	.14	.12
Concentrate:						
Table	72.53	74.38	75.38	74.53	76.52	76.04
Flotation	62.14	71.99	70.89	71.80	71.36	69.42
Combined	69.09	73.54	73.69	73.47	74.30	72.84

1 – First five months.

Table 29.–Complete analyses of table and flotation concentrates

Product	Assay, per cent										Ounces per ton,
	Pb	Cu	Insol.	Fe	CaO	MgO	S	Zn	CO_2	Ni-Co	Ag
Table concentrate	75.89	0.14	0.3	4.5	0.8	0.5	17.0	0.5	1.1	0.12	1.2
Flotation concentrate	73.03	1.11	1.6	3.0	.8	.5	15.9	2.4	1.2	.15	1.9

Table 30.–Representative lead assays of shift samples

Product	Assay, lead, per cent
Gravity concentrates of A-middling section	57.3
Gravity concentrates of B-middling section	71.1
New flotation feed	2.21
Feed to roughers	2.35
Denver Sub-A concentrates (finished flotation concentrate)	73.2
Denver Sub-A tailings	27.2
Cleaner concentrate	45.1
Cleaner tailing	6.21
No. 1 recleaner concentrate	56.9
No. 2 recleaner concentrate	49.6
Recleaner tailings (No. 3 recleaner concentrate and tailing)	4.01
Reground low-grade table middling:	
Rougher feed	6.36
Rougher tailing	0.39
Rougher concentrate	69.6
Cleaner tailing	4.42

Table 31.–Metallurgical data

	1926	1927	1928	1929	1930	1931[1]
Ore treated, total ... tons	1,551,776	1,642,945	1,523,218	1,577,351	1,473,801	492,302
Per 24 hours, average ... do.	5,022	5,301	4,929	5,177	5,047	5,076
Production per man-shift ... do.	35.34	39.31	42.05	45.87	50.38	69.93
Mill shifts operated	927	930	927	914	876	291
Time per day ... hours	24	24	24	24	24	24
Period ... do.	8	8	8	8	8	8
Concentrate, total ... tons	77,713	79,840	70,905	69,942	65,609	21,790
Rate per 24 hours, average ... do.	251.50	257.55	229.46	229.56	224.70	224.61
Table, total ... do.	52,518	51,811	44,170	42,823	37,434	11,448
Rate per 24 hours, average ... do.	169.96	167.13	142.94	140.55	128.19	118.02
Flotation, total ... do.	25,195	28,029	26,735	27,118	28,174	10,342
Rate per 24 hours, average ... do.	81.54	90.42	86.52	89.01	96.51	106.59
Recovery:						
Total lead ... per cent	91.70	94.53	95.77	95.91	96.24	96.64
By tables ... do.	67.6	64.9	62.3	61.2	57.0	52.5
By flotation ... do.	32.4	35.1	37.7	38.8	43.0	47.5
Water consumption:						
Gross ... ratio	–	–	–	12 to 1	12 to 1	12 to 1
Net ... do.	–	–	–	1 to 1	1 to 1	1 to 1

1 – First five months.

Table 32.-Screen analyses of table and flotation tailings, per cent

Size, mesh	Table tailing: Weight	Table tailing: Assay, lead	Table tailing: Distribution of lead	Flotation tailing: Weight	Flotation tailing: Assay, lead	Flotation tailing: Distribution of lead
Plus: 28	3.5	0.15	4.9	-	-	-
35	13.1	.13	15.9	-	-	-
48	28.7	.10	26.8	-	-	-
65	42.3	.09	35.4	3.6	0.13	3.9
100	8.6	.10	8.0	8.0	.09	6.0
150	3.0	.10	2.8	12.1	.06	6.1
200	.5	.29	1.4	10.6	.06	5.3
325	}			24.3	.07	13.3
Minus: 325	.3	1.70	4.8	41.4	.19	65.4
Total	100.0	0.11	100.0	100.0	0.12	100.0

LABOR

The workmen are all American-born and in most cases are natives of the district. The labor turnover is low.

A total of 69 men are employed. A superintendent and assistant superintendent are directly in charge of the plant. There are six shift bosses, one per shift in the secondary crushing plant, and one per shift in the concentrator. Sixty-one men are employed in operation and maintenance.

Including the shift bosses, three 8-hour shifts of 11 men per shift are employed in the concentrator, and three shifts of four men in the secondary and roll crushing plants. Two shifts of one man are operated in the primary crushing, and in the flotation concentrate filtering and drying plant. In the case of the 2-shift operation the shifts are separated by four hours. Three men are employed on sampling. The maintenance is handled by a crew of 15 men. The wage scale for the principal classes of labor for 1931 is given in Table 33.

COSTS

The milling costs in units of tons per man per 8-hour shift, kilowatt hours per ton run-of-mine ore, and pounds of reagents per ton of flotation feed are given in Table 34.

Table 33.-Labor costs in 1931[1]

Type of labor	Wage per 8-hour shift
Shift bosses	$4.70
Crusher feeders	3.50
Roll-floor men	3.70
Screenmen	3.25
Rod-mill men	3.70
Rod-mill helpers	3.10
Tablemen	3.70
Table helpers	3.25
Flotation men	3.90
Flotation helpers	3.10
Filter and dryer men	3.90
Concentrate loaders	3.50
Motor attendant	3.55
Maintenance:	
Foreman	4.90
Machinists	3.30 to 4.50
Machinists' helper	3.15
Mill repairmen	4.10 to 4.30

1 - First five months.

Table 34.-Summary of costs in units of labor, power, and flotation reagents

		Per man per 8-hour shift[1], tons
Labor	Dry crushing	340
	Wet grinding	567
	Table concentration	567
	Flotation	850
	Sampling	1,700
	Concentrate filtering, drying, and loading	1,700
	Superintendence	1,020
	Maintenance	283
	Overall	70

		Per ton of run-of-mine ore, kilowatt-hours: 1930	1931[1]	Distribution of power: 1930	1931[1]
Power	Dry crushing	3.4	3.4	18.9	20.4
	Wet grinding	4.2	4.0	23.3	24.0
	Table concentration	1.6	1.7	8.9	10.2
	Flotation	4.6	4.6	25.6	27.5
	Slime disposal	.4	.4	2.2	2.3
	Concentrate loading	.2	.1	1.1	.6
	Water supply	2.7	2.0	15.0	12.0
	Tailing disposal	.9	.5	5.0	3.0
	Total	18.0	16.7	100.0	100.0

		Per ton of flotation feed, pounds: 1930	1931[1]
Reagents	Cresylic acid	0.185	0.20
	or		
	Pine oil	.053	.06
	Potassium xanthate	.079	.083
	and		
	Sodium cyanide	.036	.04

1 - First five months, 1931.

I. C. 6660

November, 1932

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

DESCRIPTION OF THE PROPERTY AND OPERATIONS OF THE LEWISTON DREDGE, LEWISTON, CALIF.

BY

LAWRENCE K. REQUA

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

DESCRIPTION OF THE PROPERTY AND OPERATIONS AT THE LEWISTON DREDGE, LEWISTON, CALIF.[1]

By Lawrence K. Requa[2]

INTRODUCTION

The property being worked by the Lewiston dredge of Placer Development (Ltd.) is on the Trinity River in Trinity County, Calif., about 9 miles north of the town of Lewiston (see fig. 1). The nearest rail point is at Redding, Calif., which is exactly 50 miles by road from the dredge camp. Recently a very good section of State highway has been constructed from Redding to Weaverville, and it is now possible to use this highway for 35 miles of the distance between the dredging property and Redding. The remaining 15 miles, however, are over ordinary dirt road. No trouble is experienced in reaching the property except for a few weeks during the middle of the winter when snowstorms may make travel difficult but not impossible.

At present the property consists of about 500 acres of patented land, which extends for about 3 miles along the main Trinity River and Stuart's Fork. It is being worked with one 7-cubic foot, Bucyrus-type dredge, which has an average monthly capacity of about 100,000 cubic yards.

To the end of 1931 approximately 10,000,000 cubic yards of gravel had been dredged.

HISTORY OF OPERATION

The present dredge, which was built by the Lewiston Dredging Co., a subsidiary of the Metals Exploration Co., began operation in January, 1923. The machinery formerly used in the Valdor dredge, which operated at Junction City on the same river, was installed in this hull. The dredge was worked by the Lewiston Dredging Co. until January, 1929, at which time it was acquired by Mark L. Requa, trustee for R. F. Lewis and L. K. Requa, who operated it until August, 1930. At that time operations were taken over on a lease basis by Placer Development (Ltd.) which is working the property at present.

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6660."

2 - One of the consulting engineers, U. S. Bureau of Mines, and geologist.

Another placer operation in this area is that of the Trinity Dredging Co., whose property adjoins on the south that which was first worked by the Lewiston Dredging Co. The Trinity Dredging Co. is using a flume-type dredge. Also considerable work has been done in this area by hand and hydraulic methods, both in the present river channel and on segments of an old high channel. As the greater part of this work was done many years ago, there is no authentic record of what was produced. Judging from the area worked and the ditches installed, much of it must have been profitable.

TOPOGRAPHY

The dredge camp is at an elevation of 2,000 feet above sea level. The relief of the country on either side of the river is rough and precipitous, and the course in which the river flows is generally fairly narrow. Occasional meadows range from a quarter to a half mile wide, and it is in these meadows that the best operating results are secured. There are three of these meadows on the property, two of which have already been worked. The dredge is now entering the third and last one.

All of the dredging land is covered to a greater or less extent with timber, consisting of small pine and cottonwood, which has to be cleared in advance of the dredge.

GEOLOGY

The bedrock in this area is mostly a series of Paleozoic rocks which occur extensively in this part of the State. Locally the series consists of meta-andesite, sometimes classified as "greenstone," and slate. The greenstone constitutes the bedrock under most of the area being dredged. It is extremely hard and in many places in the narrow channel is quite rough. It does not make a very good gold catcher and as a consequence the best concentration of gold may be found anywhere from 1 to 15 feet above the bedrock. Moreover, due to hardness of the bedrock and to its roughness in places, it is very difficult if not impossible to clean it thoroughly. The slate, on the other hand, is a good gold catcher, and is easily dug.

The depth of the gravel to bedrock ranges from 10 to 45 feet. In the broader meadows the depth probably averages between 35 and 40 feet, while in the narrower parts of the river channel the dredge has on numerous occasions passed over bedrock with only a foot or two to spare.

The bank carried in front of the dredge ranges in height from 0 to 15 feet. At times, when digging in the river, the upper edge of the cut is a foot or two under water.

The pay gravel occurs in a fairly well-defined channel. At the extreme southern end of the property, first worked by the original company, this channel was 800 feet wide. Where recently worked, however, the pay streak ranges from 100 to 400 feet in width. Within this pay channel the values are spotty, which seems to be a characteristic of the Trinity River.

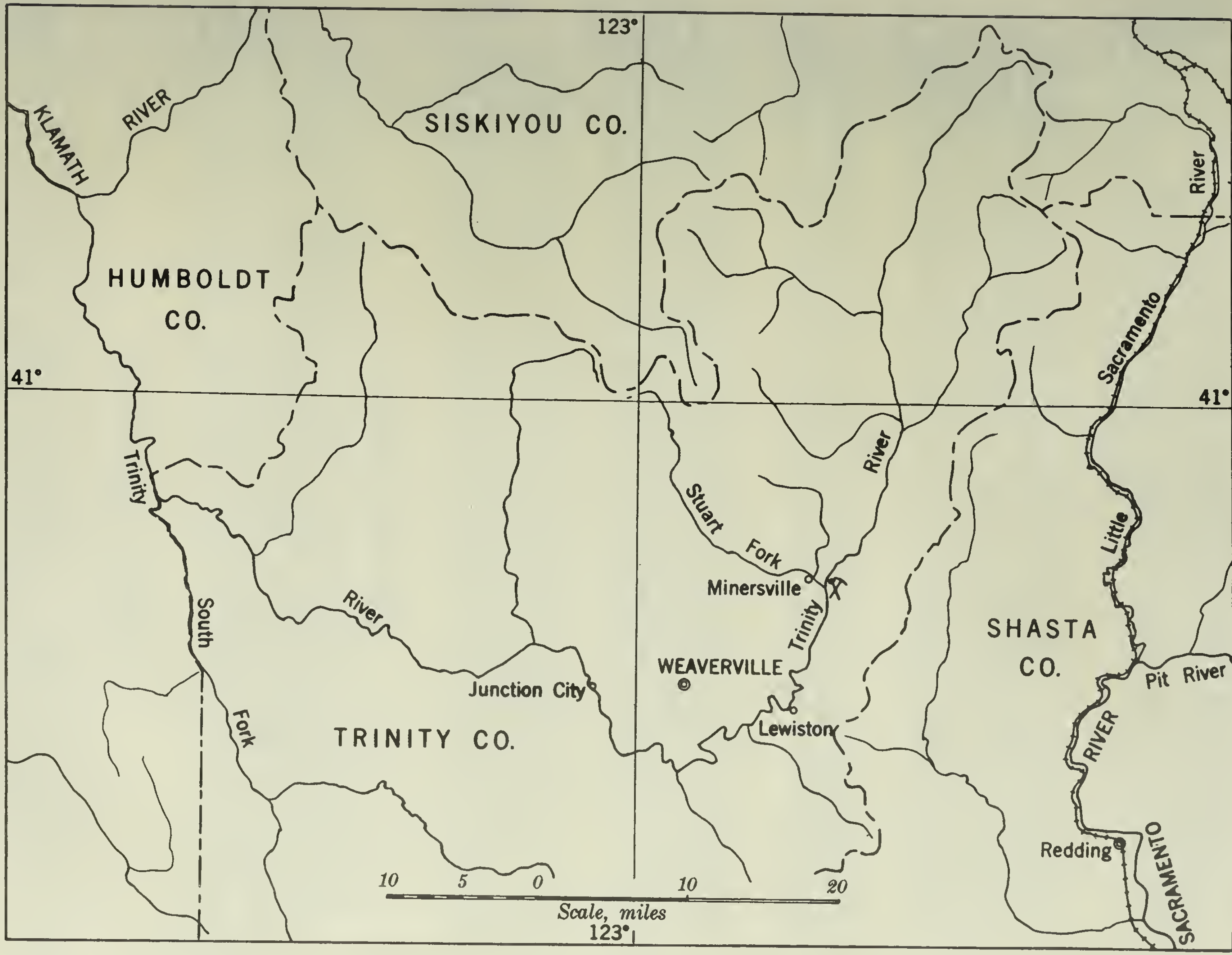


Figure 1. Northern Trinity County, Calif.

Remnants of an older channel that existed at one time in this drainage area are easily seen, as they were extensively hydraulicked by the early miners. This channel was from 20 to 30 feet above the bed of the present Trinity River and apparently was considerably richer than the present channel. It has been observed that the dredge clean-ups frequently improve when working near a segment of the older channel.

The gravel is for the most part easily dug. Large bowlders occur, but not frequently enough to cause any appreciable delay. Clay when it does occur usually is brick-red in color, but is not sticky enough to remain in the buckets or to cause any difficulty in washing. This red clay generally is found at the surface. Some light-grey bedrock clay has been encountered, but this is even less sticky than the red clay and disintegrates easily on washing.

The numerous stumps left after the clearing of the dredging area are allowed to fall into the dredge pond and will usually float around the dredge, though occasionally one fouls the bucket line and has to be removed by the crew.

The greatest difficulty in operating has been due to the occurrence of cemented gravel. This "cement" is found in lenticular patches at or near bedrock, usually in the narrower parts of the river channel, not in the deepest part of the channel but flanking it. Isolated occurrences of "cement" are encountered in the larger flat areas of the deposit but are not common there.

The water level in the dredge pond corresponds closely with the level of the water in the river even when the dredge is a considerable distance from it. The gravel is quite permeable, as denoted by this fact and also by the fact that shaft-sinking has always been difficult here because of excessive water.

CHARACTER OF THE GOLD

In the last 2-1/2 years of operation the fineness of the gold ranged from 777 to 918, with an average of about 900. The coarseness of the gold is about medium. Well-worn nuggest up to $3 in value are not uncommon; the largest one found recently was worth about $8. Under a glass a considerable number of spicules or fine particles of wire gold are often noted in the fines recovered in sampling churn-drill holes. Platinum has not been observed.

PROSPECTING AND SAMPLING

The property was sampled in three stages: (1) The southern end of the property was drilled by the Metals Exploration Co. prior to the time that the dredge was built in 1923. About 100 churn-drill holes were put down at that time. A few shafts were sunk but the greater part of the prospecting was done with a steam-driven Keystone portable rig, using standard prospecting

casing with an inside diameter of 6 inches. The holes were spaced 125 feet apart in roughly parallel rows 750 feet apart, as shown in Figure 2. (2) The upper ground was prospected by the Shasta Dredging Co. in 1922, using the same rig that was used in prospecting the ground of the Metals Exploration Co. Over 60 holes were put down in rows about 900 feet apart with the holes spaced 150 feet from each other. (3) When the present owners were investigating the possibilities of dredging from the lower to the upper ground, 27 churn-drill holes were sunk in the 8,000-foot interval between the two previously prospected areas. The rows of holes were about 750 feet apart and the holes were spaced 125 feet from each other in the rows. The same drill rig was used in this prospecting that had been formerly used.

The cost of prospecting can be stated only for this last stage of the work, which was done in 1928. The wage-scale was as follows:

	Per day	
Driller	$6,	plus board and transportation.
Panner	$6,	do.
Fireman	$5,	no board nor transportation.

The cost was $7.23 per foot. This included the salary of the engineer in charge and the entire cost of a new string of drill pipe--which, however, was not worn out in the prospecting.

SAMPLING OF THE DRILL HOLES

Logs of each hole were kept on standard log forms, which showed the number of 1, 2, or 3 size colors obtained in each pan, as well as the character of the formations penetrated, the volume of core removed as measured in a graduated pail, and the excess core, if any. Excess core, resulting from loose gravel sloughing into the hole, was compensated for in the calculations by a proportionate scaling-down of the values recovered by panning.

In the first two prospecting campaigns the Radford formula was used, with the factor 0.27. This formula may be written $V = \frac{R}{L \times f} \times 27$, in which V = average value of the gravel drilled through, in cents per cubic yard; R = total gold recovered from the drill sludge, in cents; L = total length of hole under consideration; f = a factor expressing the effective area of the cutting shoe on the casing, which in this instance, with standard 6-inch casing, is 0.27 square feet; and 27 is the factor to convert from cubic feet to cubic yards. As it was believed that the factor 0.27 gave too high results, 0.3068 was used in the last stage of prospecting. This figure is one developed by the Keystone Driller Co., and is in common use.

For the purpose of calculating the yardage and average value of the entire deposit, the latter was divided into blocks. The average foot-cent value of all the holes in each block was calculated and this value was assigned to the block. From the number of yards in each block and the average value, it was possible to calculate the average value of the entire deposit.

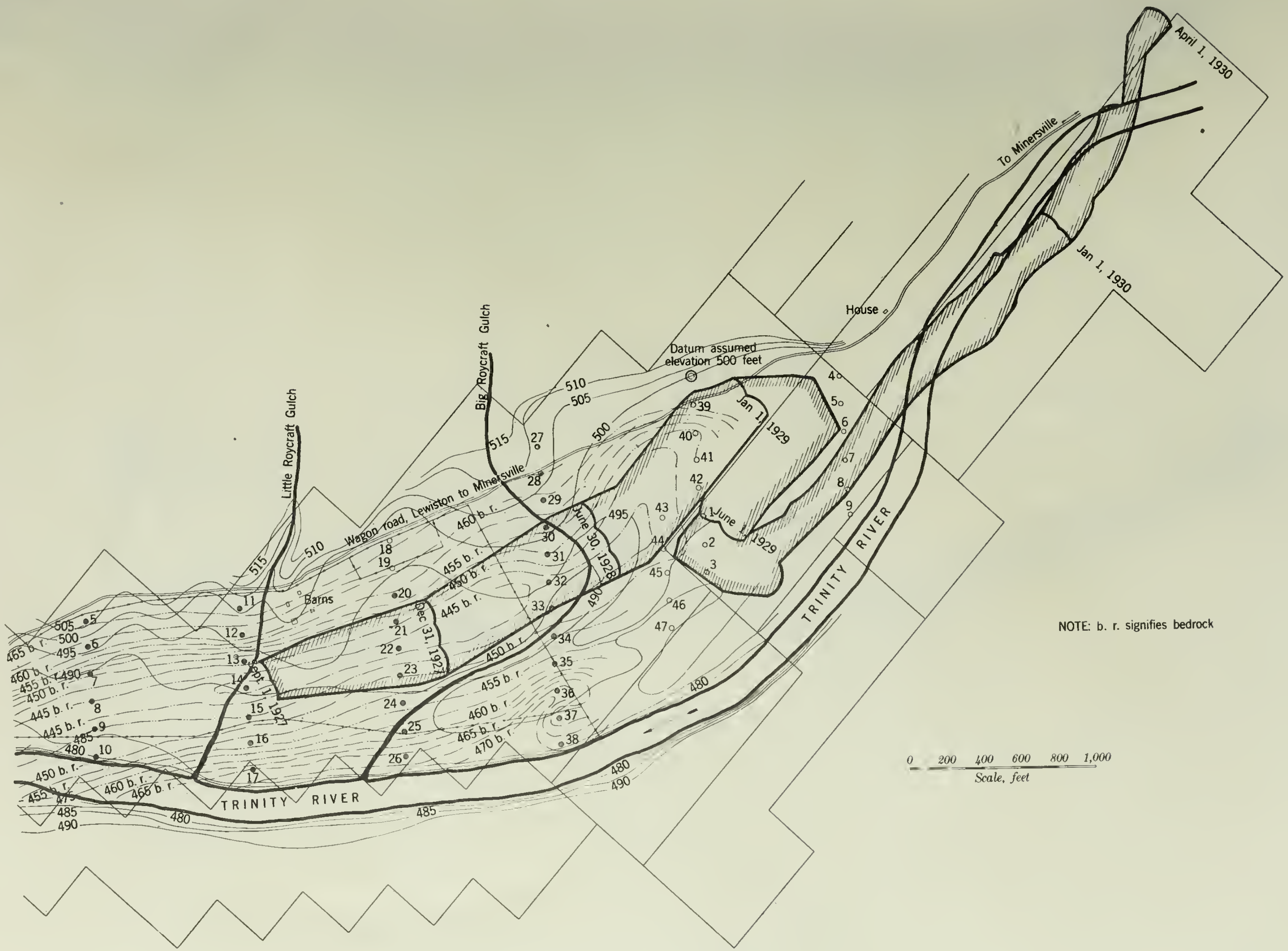


Figure 2.—Map of Lewiston dredge operations

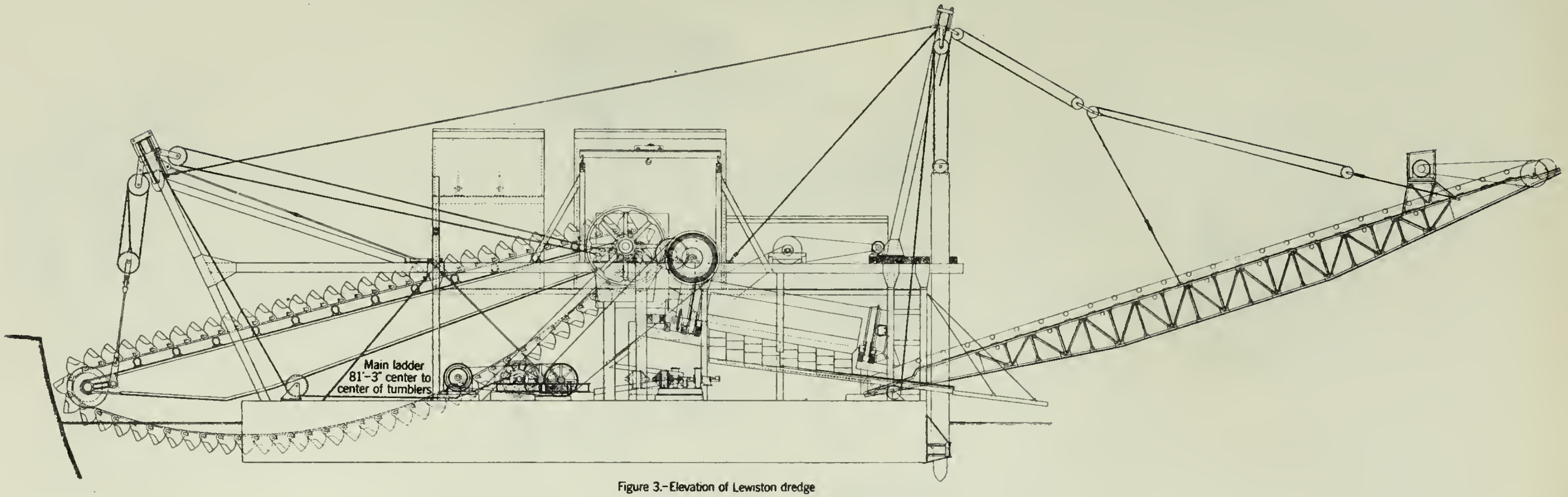


Figure 3.–Elevation of Lewiston dredge

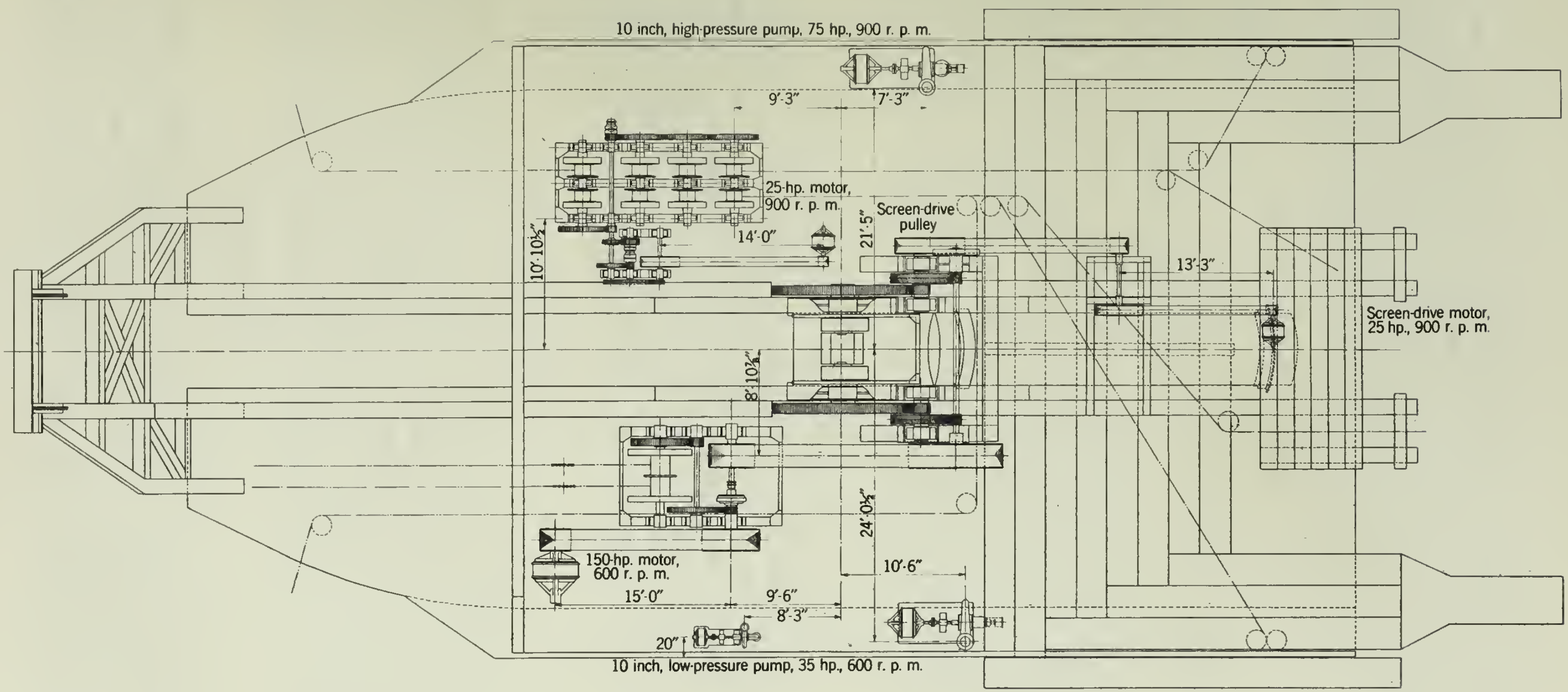


Figure 4.—Plan of Lewiston dredge

The dredge recovery to date has not equalled the estimated value. In the lower or southern ground the property was divided into two parts for the purpose of calculation. The first area showed a recovery of 77 per cent and the second 58 per cent. It is noteworthy that the first area consisted of a wide meadow and the second was a narrower river channel. Moreover, there was some slate bedrock in the first area and none in the second.

The dredge recovery for the upper area can not be stated, as dredging is just now starting there.

In the intervening area that was sampled by the present owners the dredge recovery was 62 per cent of the estimated value.

From the foregoing it may be seen that prospecting on the Trinity River in this vicinity is apt to give results that are too high, even when the pipe factor 0.3068 is used. It is probable that the low recoveries noted are due largely to the nature of the bedrock. The greenstone surface is hard and in places is rough, which makes effective cleaning of it almost impossible. Where slate or soft bedrock is found, it appears that the recovery improves. Recently a soft area in the greenstone was encountered and the recovery showed improvement.

DETAILS OF DREDGE CONSTRUCTION

The general elevation of the dredge is shown in Figure 3, and a plan in Figure 4. It is capable of digging 38 feet below water level and can carry a 15-foot bank above water line in front of the dredge, making a total digging depth of 53 feet. The total width that can be dug in one swing is slightly over 200 feet and the shortest swing that will permit progress of the dredge is about 90 feet. This is an important factor in operating in a narrow river channel.

Details of Hull

The hull, which is constructed of Oregon fir, has a length of 100 feet, a width of 43 feet, a depth of 9 feet, and a draft of 8 feet (the drawing shows a draft of 6 feet, which was the draft when operations began). The deck overhangs the hull 4 feet on either side, giving a deck 51 feet wide. The boat is 8-1/2 years old and it is believed that the hull will last about four years more. The water in which the dredge is working is always cool and this factor probably will give more than average life to the hull.

Bucket Line

The digging ladder measures 81 feet 3 inches from tumbler center to tumbler center. The lower one-third is of the closed girder type and the upper two-thirds of the open girder type. The lower tumbler is of manganese steel. It is round and is cast in two halves. The tumbler shaft is hollow, and through it passes a tie-rod which holds the ladder points together.

The upper tumbler is hexagonal, and the most noteworthy feature is that wearing plates of ordinary mild steel are used. Formerly Mayari steel plates were used, but it was found that due to their extreme hardness and the inability to seat them perfectly to the tumbler faces, they frequently would crack. Also there was considerable slippage of the bucket line when hard digging was encountered. L. Picard, chief mechanic at the dredge, suggested the use of the mild steel plates. This was a radical step, as it was thought generally in dredging circles that mild steel would not stand up, also that it would flow under the pressure of hard digging. The plates installed were of ordinary mild steel and were 2 inches thick. They were bolted and welded into place. The results have been more than satisfactory, as the average life of each wearing plate has been from four to six months and there has been no sign of flowage under pressure from hard digging. The plates seat well to the tumbler faces, the tucket line runs more smoothly and quietly than formerly, and the buckets slip and wear less.

The lower tumbler is lubricated through a grease pipe extending along the side of the ladder, by means of a high-pressure grease gun. The rollers, weighing 742 pounds each, are lubricated by grease cups.

There are 72, 7-cubic foot buckets in the bucket line, each weighing over a ton, as follows:

Part	Weight, pounds	Material
Bucket	1,545	Manganese steel
Lip	325	Do.
Bushing set	26	Do.
Pin	147	Mayari steel

The average life of a bucket lip is 8 to 12 months. After considerable experimenting, it has been found possible to weld the bucket eyes successfully after signs of fracture appear.

The digging ladder is driven by a belt-connected, variable-speed, 150-hp. motor. The ladder-hoist winch is likewise geared to this motor, but in such a way that the hoist can be run independently of the bucket line.

The movements of the dredge are controlled by an 8-drum winch, belt-driven by a 25-hp., variable-speed motor. From this winch run the four bow and stern swing lines, and the lines to raise or lower the spuds.

The main drive and winch motors are operated through controls in the front of the pilot house. The screen drive and stacker drive are similarly connected to controllers in the rear of the pilot-house. The pump switchboards are situated on the deck close to the pumps.

The power cable comes on board at the stern on the port side, where the main oil switch is placed. Thence it passes up the port side to a bank of transformers opposite the pilot house.

DREDGE OPERATION

As indicated before, the dredge is operating in a relatively narrow river channel and a large part of the dredging is done carrying only one cut (about 200 feet); however, in the wider meadows three cuts have been carried with a total width of over 400 feet. Frequently one side of the dredge pond is open to the river and as a consequence no pump is needed to supply water to the pond.

The actual capacity of the dredge varies from 90,000 to 120,000 cubic yards per month, with an average of about 100,000 cubic yards. The character of the digging is primarily responsible for the variation. If bedrock is unusually hard and rough or if cement is encountered it cuts down the yardage materially. In a few parts of the dredging area an escessive amount of sand and clay overburden was encountered, which necessitated dry washing for short distances. Dry washing is screening without water, which causes most of the gravel to pass through the screen as oversize.

Dredging in a narrow river gorge presents problems which are unknown to operators who are working large flat areas. One of the most difficult problems is to so plan the progress of the dredge that it will not be necessary either to cross the river or to work unprotected in the river during the winter months when high water and sudden freshets are likely to occur. To date it has been possible to meet this problem successfully, and as the dredge is just through the last narrow portion of the river no more difficulty from this source is expected, but for all operators considering properties on similar rivers it is a factor that should be given due consideration.

As previously stated, at times excessive amounts of sand and clay in the overburden have caused trouble. During one stage of the operation when the dredge was digging from the main river into a high sand bank, the fines going through the sluices were so excessive that the stern of the dredge grounded. As the rear of the dredge pond was open to the river, it was necessary to construct a gunny-sack dam and raise the water in the dredge pond about 1 foot. Dry washing was then resorted to until the dredge was far enough ahead to block effectually the rear of the pond with tailings. The level of the pond was then raised slowly and as the proportion of fines decreased, which it fortunately did, it was possible to resume normal digging.

Under ordinary circumstances an average of about 90 per cent running time is maintained. There have been a few big repair jobs in recent years that have pulled this percentage down considerably, but with nothing but the ordinary routine repairs the above running time is sustained. There have also been times when the physical characteristics of the deposit have caused considerable lost time. No great amount of trouble has been caused by bowlders. Those brought up in the bucket line, which the winchman believes are too large to go over the stacker, are removed by the crew on the forward deck. If a bowlder is encountered which is too large for the bucket line to bring up, it is worked over to one side of the pond.

Of the total time lost, about a fifth is charged to clean-up operations, and an equal amount to moving and other work on the shore lines which control the movement of the dredge. An analysis of lost time in the first six months of 1930 is as follows:

Total working hours	3,115
Hours shut down	438
Digging time, per cent of total working time	86

Cause of delay	Per cent of shut-down time
Bowlders	0.2
Buckets	4.7
Clean-up	17.0
Hopper	3.0
Ladder	2.3
Lines	20.0
Motors	5.5
Power lines	1.7
Power off	3.4
Pumps	0.7
Screen	7.4
Stacker	9.7
Stepping up	3.9
Spuds	1.6
Tables	1.4
Tumbler, lower	1.2
Tumbler, upper	0.8
Winches	6.0
Miscellaneous	9.5
	100.0

During the above period the dredge was shut down from January 13 to 17 because of cold weather, the chief difficulty being the freezing of the stacker. On March 4 a 12-hour shutdown resulted from high water in the river which made dredging dangerous. From May 9 to 17 the dredging ceased while a dam was built to raise the water in the pond, for reasons noted elsewhere. As this was an exceptional occurrence, and will not happen again, it was omitted from the above analysis.

A printed form of daily shift report (fig. 5) is filled out for each shift. This shows the men on shift, the hours each has worked, and the length of each shutdown and the cause. The men sign their names, with their time and overtime if any, and this is checked and signed by the dredge master. The height of bank at several places across the cut as well as the depth below water level that the dredge is digging are noted. Two types of depth indicators are used. One is an automatic recording device and the other a visual one consisting of a moving arrow on a graduated scale. The automatic record checks the length of shutdowns and the depths recorded by the winchmen from

the visual one, but for purposes of yardage calculation the depths taken from the visual depth gage are used.

Yardage is calculated on each clean-up day for the period since the previous clean-up. A map on a scale of 50 feet to the inch is used to record the progress of the dredge. With transit and stadia rod the dredge cut is surveyed and then is plotted on the map. The area dredged is measured with a planimeter and by multiplying this by the average dpeth for the period covered, the total yardage is obtained.

GOLD-SAVING EQUIPMENT

The discharge from the main hopper goes directly to the screen, which is 30 feet 7 inches long, 6 feet in diameter, and set at 10 degrees from the horizontal. There are four sections to this screen, each section containing six manganese steel screen plates. The upper two sections of plates have 3/8-inch holes and the lower two have 1/2-inch holes. Obstruction rings of manganese steel are located between each section of screen plates and manganese steel lifter bars are placed between each screen plate. The screen is driven through its two forward 41-inch rolls by a belt-connected 25-hp. motor. It rotates in a clockwise direction, looking at it from the stern. The fines from the screen are discharged into the distributor, the trough of which is 12 inches wide. This in turn discharges onto a single bank of tables having eight sluices on each side. A high-pressure spray pipe extends the entire length of the screen and is directed slightly to the port side of the interior of the screen. Spray nozzles are also located at the openings of the distributor onto the tables.

The "save-all" sluices, as indicated in Figure 3, are on the deck directly below the upper tumbler and just back of the ladder well. The drip from the buckets falls upon a bar grizzly about 7 feet long and having a forward slope of approximately 40°. The oversize drops into the ladder well. The undersize is conducted to the forward end of a sluice box, about 9 feet long. This discharges into another box directly beneath but about 13 feet long and pitching forward so that it empties into the ladder well. Both boxes slope 1-1/4 inches in 12.

The transverse sluices are eight in number on each side of the dredge. Each sluice is 2 feet 7-1/2 inches wide and is set on a grade of 1.5 inches per foot. The transverse sluices discharge into four tail sluices, the outermost of which is outside the dredge housing. The other three are inside the housing and discharge through a 3-way tail sluice regulator, whereby the tailings can be either directed to port, to starboard, or directly astern as the occasion for spud ground demands. The tail sluices are likewise set on a grade of 1-1/2 in 12.

Hungarian-type riffles are used in the sluices. These consist of transverse wood strips 1 inch wide and 1-1/4 inches high, but beveled on top so that the upstream edge is only 1-1/8 inches high. Each is capped with a 1/4-

inch mild steel strap, 1-1/4 inches wide, which is set flush with the front edge of the riffle and projects 1-1/4 inch back of it. The side members are 1-1/4 by 1-1/4 inch strips.

Water for gold-saving operations is supplied by two 10-inch centrifugal pumps (Krogh, No. 10, type B, form E). One is a high-pressure pump, driven at 900 r.p.m. by a direct-connected, 75-hp. motor. This pump is on the starboard deck overhang just opposite the forward end of the screen. It has a capacity of 3,200 gallons per minute, taking water from the dredge pond through a short vertical-suction pipe and supplying it direct to the 10-inch header and the 15, 1-1/4-inch spray nozzles inside the screen. The other main pump is on the port deck overhang, likewise abreast of the forward end of the screen. It is a low-pressure pump, having a capacity of 3,000 gallons per minute, and is driven at 600 revoluations per minute by a 35-hp. motor. It is connected to the two 7-inch headers and the 16, 2-inch pipe nozzles which supply water to the distributor outlets on either side of the screen. A 4-inch centrifugal pump is located on the deck overhang just ahead of the low-pressure pump. This is driven at 1,200 revolutions per minute by a 5-hp. motor. It serves as both primary pump and bilge pump, having pipe connections to four bilge sections at the four corners of the hull, and overboard discharge and suction. This pump and both the large pumps are inter-connected by a 4-inch line, which likewise branches to hose connections at several points about the dredge.

The stacker belt is 32 inches wide and is 7-ply, 32-ounce duck with 5/16-inch top covering including the breaker strip and a 1/16-inch bottom covering. It is the practice to calculate the life of these belts in terms of the yardage carried and it is found that the average belt will carry slightly in excess of 1,000,000 cubic yards before it is worn out. The troughing rolls on the stacker are 3 feet 8 inches apart. Two bowlder retarders are situated, one about a third and the other about two-thirds of the way up the stacker, each having six steel fingers. These prevent bowlders from rolling down the stacker. The stacker is driven at a speed of 380 feet per minute by a 30-hp. motor which is housed at the upper end of the stacker. This end usually is about 37 feet above the level of the pond.

GOLD-SAVING OPERATION

There is nothing distinctive or unusual in the operation of the gold-saving plant. There is not enough clay in the ground to prevent the buckets from dumping their entire load, and as a consequence there is no trouble in washing the gravel clean in the screen. No attempt has been made to estimate the percentage of material discarded by the screen, so that it is impossible to state what this is.

After each clean-up the dredge is run until the riffles are fairly well filled up. Digging is then stopped temporarily while quicksilver is scattered on the upper part of the sluices in amounts that experience has shown to be sufficient.

No exact calculation of the amount of gold recovered in each stage of the operation has been made, but a very large per cent is known to be recovered in the upper half of the first two transverse sluices.

Ordinarily, clean-ups are made every 10 days. At every clean-up all of the riffles in the transverse sluices are removed. Each sluice is divided into an upper and lower part by two unremovable riffles. The contents of the sluices are worked down against these unremovable riffles with small hand hoes, and a considerable amount of the material is sluiced off. The remainder is scooped up and carried to the superintendent, who "boils it down" in a pail with the aid of a stream of water. The quicksilver and amalgam remain in the pail and the black sand is "boiled off." This is collected and put into sacks and carried to the gold house where it is run through a long tom containing a quicksilver trap. Lately long toms have been installed on the dredge and this part of the operation has been done there. The quicksilver and amalgam remaining in the pail and that in the trap of the long tom are placed in a canvas strainer and the quicksilver removed, the amalgam remaining in the strainer. The black sand remaining is saved up and run through an amalgam barrel about once in six months.

The amalgam is placed in a "bleeder," which is nothing more than a metal funnel about a foot high, constructed so that it will fit over a receptacle to catch the quicksilver that drains off. This operation has been found to shorten materially the time necessary to retort.

After the amalgam has been sufficiently "bled," it is placed in a retort and the quicksilver driven off, following which it is placed in a crucible, the temperature raised somewhat, and the gold melted. It is then poured into a mold.

The tail sluices are not cleaned up on every clean-up day and are sometimes not touched for a month. The save-all is cleaned up once a month.

As a precaution against theft all of the sluices are inclosed with heavy wire screening and are kept under padlock at all times. Full insurance is kept on the clean-up from the time it is collected in the form of amalgam on the dredge until it reaches the smelter.

REPAIRS

Routine repairs usually are made on clean-up day when the dredge is shut down. Every repair that would require the dredge to be shut down is anticipated, if possible, and everything is in readiness at this time, so that no further shutdowns and loss of running time will be incurred.

A machine shop is maintained at the dredge camp and all of the repairs, with very few exceptions, are accomplished on the property.

STREAM POLLUTION

Until this year (1931) it was not necessary to give any thought to stream pollution, due to the fact that the Trinity River, throughout its entire

course, flows through a region in which agriculture is possible at only a few limited localities. None of the antidebris legislation that was in effect in other parts of the State applied to the Trinity River watershed.

Certain resort owners, mostly in Humboldt County, advanced the claim that the pollution of the Trinity River by the hydraulic miners and the dredges during the fishing season was detrimental to their interests. The controversy was carried to the State legislature, where after hearing the evidence of both sides a compromise bill was enacted. This bill provides that during three months in the summer, starting on July 15 of each year, the sediment in the river must not exceed 50 parts per million, by weight, at a point 1 mile below the Trinity - Humboldt County line (see fig. 1). It is thought that the placer properties will be able to continue operations through this period if considerable care is exercised, but as the first full season under this restriction has not gone by, it is impossible to say definitely.

POWER

Electric power is supplied to the dredge by the Pacific Gas and Electric Co. It's Redding-Eureka power line runs through Lewiston, from where a branch line runs up the Trinity River as far as Minersville. The power is supplied from the Redding end, except in case of trouble, when a small stand-by plant, located at Junction City approximately half-way between Redding and Eureka, is used.

Current is supplied at 60,000 volts to the dredge substation, where it is stepped down to 2,200 volts. Transformers on the dredge step this down further to 440 volts for use in the various motors. The power line is carried across the dredge pond on two floats, each made of four large oil barrels on a wooden frame.

No. 6 copper wire is used to carry the current from the dredge substation to the dredge. It was found that the greatest distance at which the dredge could work satisfactorily from the substation was about 1-1/4 miles. Beyond this distance the voltage was too low for efficient digging.

The power cost schedule that the dredge has been operating under recently is as follows:

Demand charge:

First 200 kw. or less of maximum demand	$300.00	per month
Next 300 kw. of maximum demand	$1.00 per kw.	per month
Next 500 kw. of maximum demand	.75	do.
All over 1,000 kw. of maximum demand	.60	do.

Energy charge (to be added to demand charge):

First 150 kw.h. per kw. per month	0.8 cent	per kw.h.
Next 150 do.	.6	do.
All over 300 do.	.5	do.

Demand: The maximum demand in any month will be the average kilowatt delivery of the 30-minute interval in which the consumption of electrical energy is greater than in any other 30-minute interval in the month. The maximum demand on which the demand charge and energy block will be based will not be less than 50 per cent of the greatest maximum demand occurring during the 11 preceding months.

The maximum demand in recent years has been 391.7 kilowatts. The actual cost for power figured on recent operations is 0.89 cent per kilowatt-hour, and the consumption of power per cubic yard of gravel dug was 1.65 kilowatt-hours for the same period.

The following is a tabulation of the electric motors on the dredge:

Motors	Speed	Horsepower	Revolutions per minute
Main driver	Variable	150	600
Swing winch	do.	25	900
Low-pressure pump	Constant	35	600
High-pressure pump	do.	75	900
Screen drive	do.	25	900
Bilge pump	do.	5	1,200
Stacker drive	do.	30	900
Shore pump	do.	25	-----

WATER SUPPLY

All water for dredging purposes is taken from the river. As the dredge is frequently working with at least one side of the pond open to the river, no shore pump is needed a good part of the time. At times when the dredge pond is not open to the river, water is supplied by a centrifugal pump direct-connected to a 25-hp. motor. At no time has it been necessary to convey water more than a quarter of a mile to elevate it more than 25 feet.

CREW

The dredge crew organization is as follows:

Superintendent
1 dredge master
1 chief mechanic
1 blacksmith
1 carpenter
3 winchmen
3 motormen
3 oilers
1 teamster

At times an extra man is employed on the day shift.

WAGE SCALE

The wage scale in the latter part of 1931 was as follows:

Dredge master	per month	$220.00
Chief mechanic	do.	250.00
Winchmen	per day	5.50
Motormen	do.	4.50
Oilers	do.	4.25
Blacksmith	do.	6.00
Carpenter	do.	5.50
Teamster	do.	5.00

ACCIDENT PREVENTION

Regular inspections of the property are made by the State safety engineer and his recommendations are followed as to safety features. Compensation insurance is carried with the State fund.

COST PER CUBIC YARD

Period covered: Jan. 1, 1929, to Oct. 1, 1931.
Cubic yards handled: 2,982,204.
Total operating cost for period: $241,396.29.
Cost per cubic yard (33 months): $0.0809.

The cost for the first 19 months was $0.0840, whereas the cost for the last seven months has been $0.0652. The principal reasons for the variation in costs were: (1) Certain physical difficulties in dredging during the early period; (2) several extensive repairs necessary in early period; (3) lower commodity prices during latter period.

For the year 1929 the cost per cubic yard was $0.0797, which was segregated as follows:

Labor	$0.0255
Superintendency	.0033
Supplies	.0325
Power	.0137
Office expense	.0003
Insurance	.0032
Taxes	.0012
	0.0797

LEWISTON DREDGE

SHIFT REPORT

SHIFT FROM________________ TO________________ DATE________________

	TIME OF SHUT DOWN						TOTAL	
	From	To	From	To	From	To	Hours	Minutes
Stepping Up								
Buckets								
Bucket Drive								
Clean Up								
Hopper								
Ladder								
Lines								
Lower Tumbler								
Upper Tumbler								
Motors								
Power								
Power Line								
Pumps								
Spray Pipes								
Screen								
Stacker								
Spuds								
Swing Winch								
Main Drive Winch								
Miscellaneous								
Remarks								
						TOTAL		

Cut Depth Below Water Bank

	Shift	Overtime	Total
Winchman			
Motorman			
Oiler			
Extra Man			
Shore Man			

Dredge Master.. Average Depth ....

Figure 5.— Daily shift report

I. C. 6661 November, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MINING METHODS AND COSTS AT FRESNILLO, ZACATECAS, MEXICO

BY

A. Livingston

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AND COSTS AT FRESNILLO, ZACATECAS, MEXICO[1]

By A. Livingston[2]

INTRODUCTION

This paper on mining practice at the Fresnillo silver mines is one of a series being prepared by the Bureau of Mines on mining practices, methods, and costs in the various mining districts of North America.

Two classes of ore are produced at Fresnillo. A low-grade oxidized silver ore; and a complex-base sulphide ore containing gold, silver, lead, zinc, and copper. This paper is confined to a discussion of operations relating to the production of oxidized silver ores. To date, more than 10 million tons of this ore have been mined. The present (June, 1931) daily production is 2,000 tons of ore, and approximately 600 men are employed in the mining operations.

ACKNOWLEDGMENTS

Thanks are due to Thomas C. Baker, general manager of The Fresnillo Co., for free use of his article on Fresnillo glory-hole practice,[3] and for many helpful suggestions in the preparation of this paper.

LOCATION

Fresnillo is 35 miles north of the city of Zacatecas, Mexico, and 750 miles south of El Paso, Tex. It is connected with the Mexican Central Railroad by a 5-1/2-mile standard-gage spur owned by the mining company. The town, of 12,000 population, is located in a broad semiarid valley on the Mexican plateau, at an altitude of 7,300 feet above sea level. It is partly surrounded by a number of low hills, the largest of which, called

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6661."

2 - One of the consulting engineers, U. S. Bureau of Mines, and assistant mine superintendent, The Fresnillo Co.

3 - Baker, Thomas C., Fresnillo Glory-hole Mining Practice: Eng. and Min. Jour.-Press, vol. 116, Dec. 1, 1923, pp. 931-942.

Proaño Hill, lies to the south. In and under Proaño Hill lies an extensive deposit of silver ore which has been the basis of important mining operations for the last 350 years and which, under the present more intensive exploitation, still has a long productive life before it.

HISTORY

The Spanish "conquistadores" reached Zacatecas in the year 1546 and discovered the enormous bodies of silver ore that have placed the Zacatecas district fourth in the history of the world in total production of silver bullion. Though only 35 miles from Zacatecas, the Fresnillo mines were not discovered until 1570. Mining operations begun at that time were carried on intermittently and through many vicissitudes for 250 years. By 1835 most of the richer and easily accessible ore had been exhausted, and the mines had reached a depth where drainage of the workings had become a vital problem with which the many small companies and independent operators were individually unable to cope. The various properties were consolidated by government decree and control of the whole operation passed to an English company.

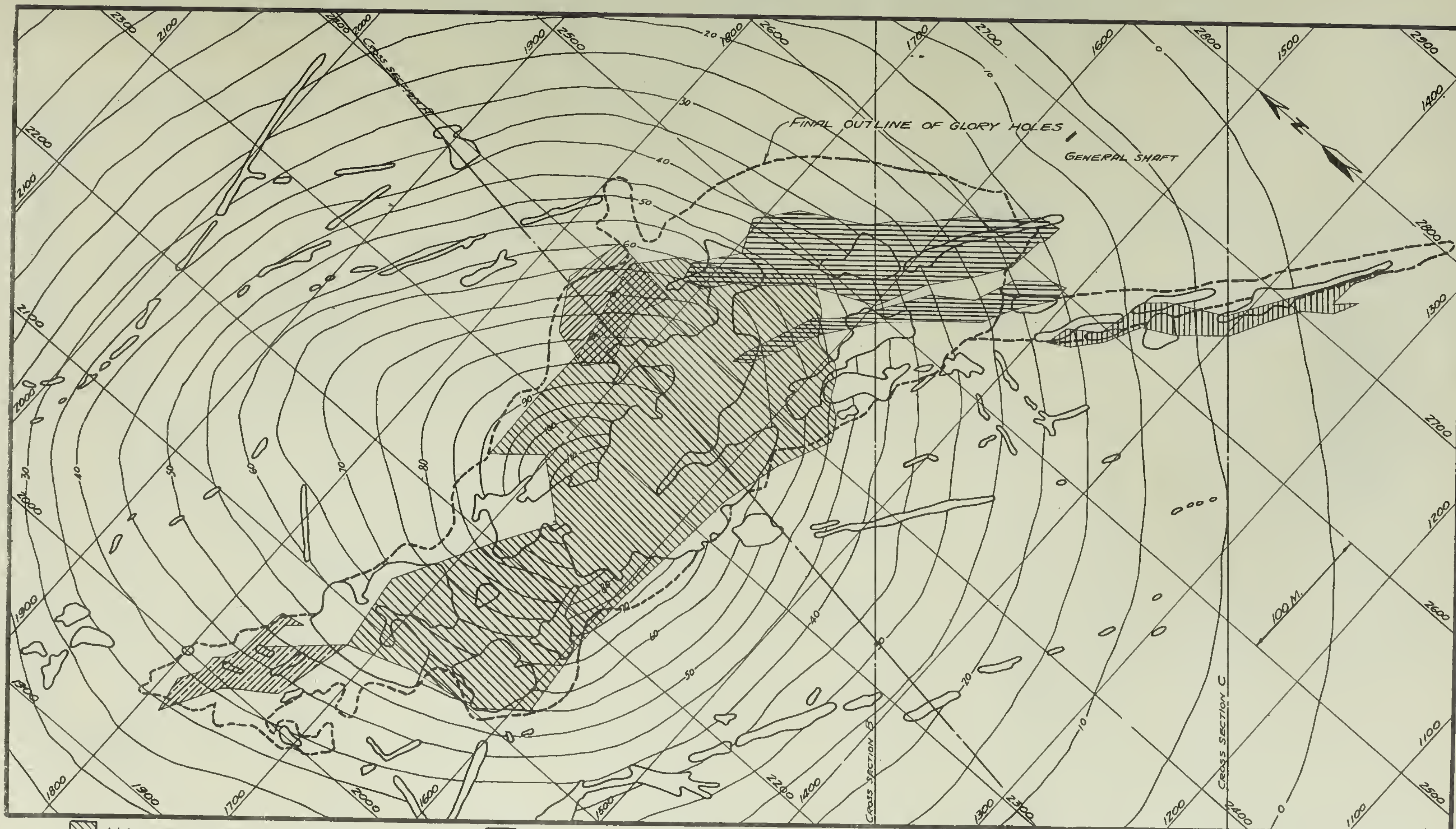
For the ensuing 30 years the mines experienced a tremendous revival under the influence of an able and progressive management. Two Cornish pumps were brought from England and hauled overland on wagons from Vera Cruz to Fresnillo. They were erected in a very substantial way, with surface buildings, boiler houses, and stacks of cut stone, the latter about 80 feet high.

Two shafts were sunk, one 11 by 19 feet, the other 10 by 14 feet, and reached a total depth of 1,400 feet below the plain. Mining operations on a large scale for the period were successfully carried on at that depth.

During the period, the Patio process for extracting the silver from the ore by amalgamation was brought to a high point of development. Steam-driven arrastres, colloquially called "tahonas," were used to grind the ore. The Hacienda Proaño was constructed, 1,150 feet square and enclosed in walls 18 feet high and 3 feet thick; also a theatre, a church, and a large building to house a School of Mines. All of this construction was done on an impressive scale and in a very substantial manner. The old buildings still survive and are in use, after a hundred years.

Operations on this scale finally ceased in 1867. The immediate cause for the shutdown was inability to secure necessary supplies, chiefly fuel for the boilers, during the period of political and economic disruption following the French occupation of the country. The water level in the mines rose rapidly after the shutdown. From 1867 to 1903 mining operations were of small importance, and consisted of robbing pillars of richer ore above the water level and re-sorting old dumps on the surface.

The Fresnillo Mining Co. entered the district in 1903 and erected a lixiviation plant for the treatment of old tailings. A few years later it extended activities to the mine and in 1911-12 erected a 500-ton cyanide plant to treat the low-grade ore available in dumps and surface workings on



HILLTOP OREBODY, EL. + 50 M.
SAN NICOLAS EXTENSION, EL. + 30 M.
SAN PEDRO OREBODY, EL. 0 M.
CATILLAS OREBODY, EL. - 44 M.
PILAR, EL. - 105 M.
OUTLINES OF SURFACE WORKINGS

Figure 1.- Plan of orebodies

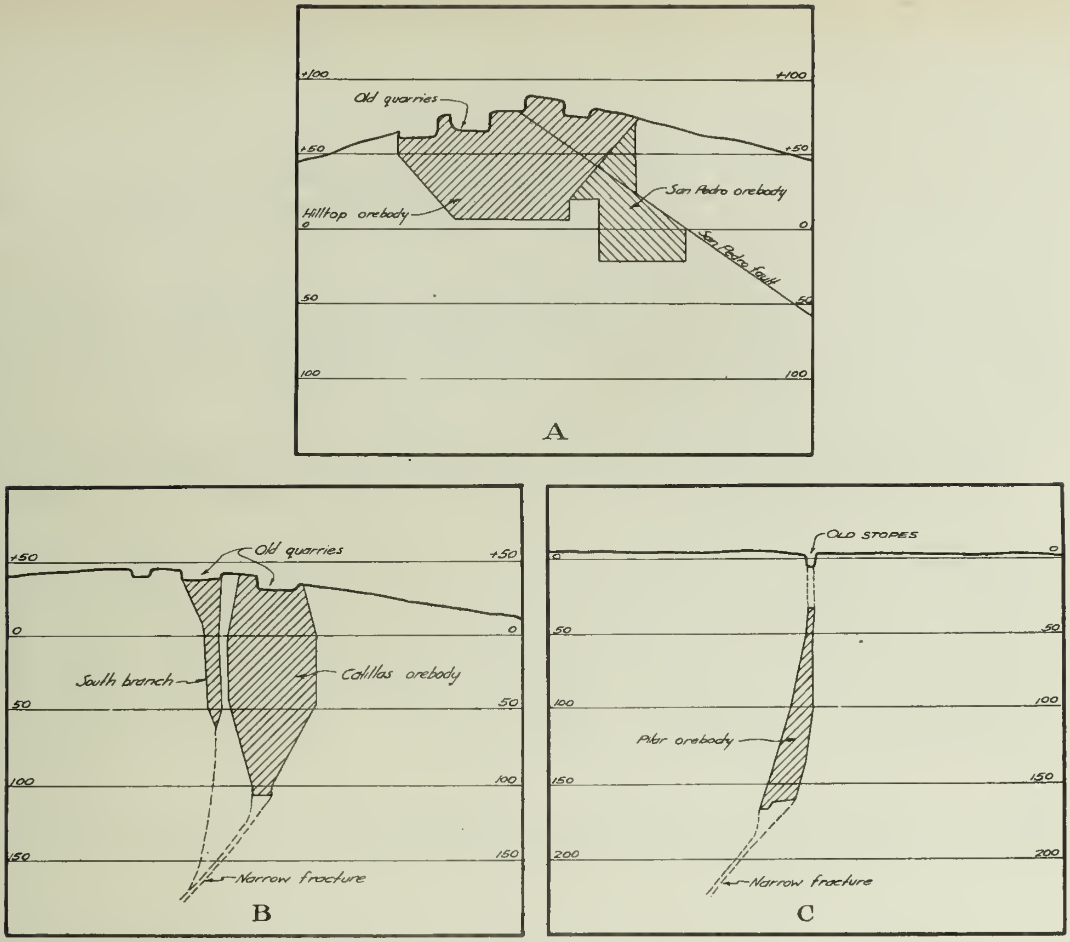


Figure 2.- Cross sections of orebodies: A, Hilltop and San Pedro, looking west; B, Catillas, looking northwest; C, Pilar, looking northwest

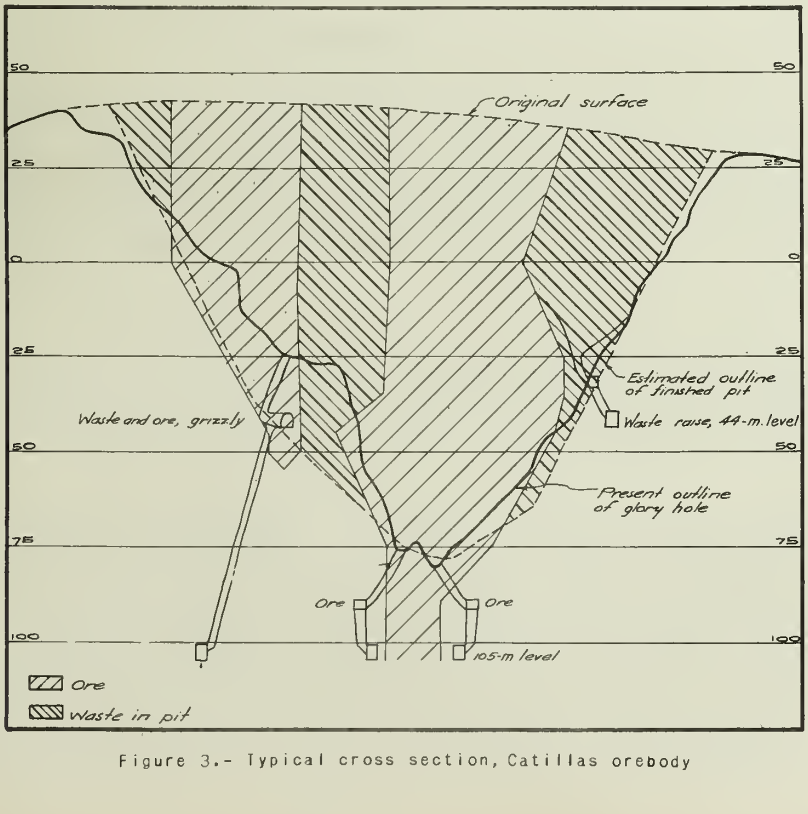


Figure 3.- Typical cross section, Catillas orebody

Proaño Hill. Operations had to be suspended in 1913 because of a revolutionary upheaval and were not resumed until 1919.

In November of that year the property was leased to the Mexican Corporation, an English company, and a modern, large-scale operation was undertaken. The basis of this operation was the Hilltop orebody, proved by the exploration work of the Fresnillo Mining Co. to contain at least 5,000,000 tons of payable ore. A modern cyanide plant having a capacity of 2,200 tons per day was built to treat this ore. The plant was subsequently enlarged and has treated a maximum of approximately 125,000 tons of ore per month. Thorough exploration of the property added greatly to the ore reserves.

Only very meager data exist as to the production of former operators. However, on the basis of the old tailings left by them, it appears reasonable to conclude that the mine had produced, prior to the advent of the present company, a total of at least 4,000,000 tons of ore. From this at least 20 ounces of silver per ton was recovered, making a total production of at least 80,000,000 ounces of silver. These figures are approximately and probably low.

Since the inception of its operations the present company has produced 10,046,000 tons of ore from which 41,275,000 ounces of silver has been recovered.

Total production from the mines is thus shown to be in excess of 120,000,000 ounces of silver. Gold occurs in the ratio of about 1.4 grams per kilo of silver, or 0.14 per cent.

GEOLOGY

Proaño Hill is dome shaped, rising with gentle slopes to about 325 feet above the plain. Its major axis lies east-west and is about 4,500 feet long. Transversely, its minor axis is about 3,000 feet long. The plan (fig. 1) shows the surface of the hill, the principal surface workings, and the relative locations of the principal orebodies. Sections of the orebodies are shown in Figure 2.

The country rocks are shales and graywackes, probably variants of the same sedimentary series, of Devonian age. No igneous rocks have been positively identified within the developed area, although certain minor variants of the graywacke have been petrologically classified as rhyolite tuff.

The orebodies in Proaño Hill constitute a true stockwork formed in an extensive zone of intense fracturing. On several of the larger fractures the movement, while small, was sufficient to make formal (regular) veins. Faulting of importance was entirely absent. Because of the friable nature of the formations, cross fracturing and shattering took place in every conceivable direction, and the entire fractured zone was subjected to the mineralizing influences. Except in a few of the larger fissure veins the deposition of minerals took place in minute veinlets on the fracture planes

with little penetration or alteration of the intervening masses. In many superficial aspects the whole deposit bears a resemblance to the porphyry-copper deposits, though, of course, it is fundamentally different from them.

There is very little segregated quartz in the orebodies. Nevertheless, Proaño Hill owes its existence to a pronounced degree of silicification of the country rocks, which very probably was from the same source as the metallization. The principal metallic mineral introduced was pyrite, accompanied by traces of lead, zinc, and copper sulphides, and carrying also the original silver minerals. Manganese in some form was a very important constituent of the mineralization. There is little evidence of secondary sulphide enrichment. The entire stockwork has been thoroughly oxidized, and the ore has a characteristic reddish to blackish color.

The higher oxides of manganese tend to reduce the solubility of the silver in cyanide, and have presented a difficult metallurgical problem. Mill recoveries have averaged between 70 and 75 per cent.

Again referring to Figures 1 and 2, it is noted that whereas individual names have, for convenience, been applied to different parts of the deposit, as "Hilltop," "Catillas," "Pilar," etc., the whole may be regarded as a single continuous orebody of irregular form. From the top of Proaño Hill to the lowest part of the Pilar orebody the vertical extent of the deposit is approximately 850 feet.

PHYSICAL CHARACTERISTICS OF ORE AND ENCLOSING ROCKS

Both the stockwork and the enclosing rocks have three characteristics which are favorable to mining operations and which have had an important influence in securing low mining costs.

1. The rocks drill very easily, except in a few minor areas of more intense silicification.

2. The ground stands remarkably well and a comparatively small amount of timbering is required to support the workings.

3. The rocks are friable and readily break into comparatively small pieces under the impact of high explosives.

Hilltop Orebody

The form and extent of the Hilltop orebody are shown in Figures 1 and 2,A. The outstanding features of the orebody that affected the choice of a mining method were:

1. It was comparatively shallow.

2. Its greatest horizontal extent was at or near the surface.

3. The silver content of the ore varied considerably in different parts of the deposit, so that many points of attack would be necessary to maintain a balanced production.

The shape and position of the orebody indicated the choice of an open-cut mining method. Steam shoveling and glory holing were considered, and the latter method was selected for the following reasons:

1. Glory-hole mining is extremely simple in detail, and could be carried out by the untrained labor available without assistance of the large foreign organization that would have been required by the steam-shovel method.

2. To secure comparable costs, large steam-shovel units would have been required, and the orebody could have been attacked in only a few places, whereas with the glory-hole method many points of attack would be available and a balanced production could be more easily maintained.

A reserve of broken ore could be maintained in the glory holes to assure a constant milling rate, whereas any serious breakdown in shovel operations would entail an immediate reduction in milling rate.

4. The preliminary expense for equipment and preparation was greatly in favor of the glory-hole method.

San Nicolas Orebody

The location and extent of the San Nicolas orebody will be apparent upon reference to Figure 1. It constitutes a westerly extension of the Hilltop orebody, lying along the principal fractures under the San Pedro fault, and in an area of intense cross fracturing.

Apart from the principal San Nicolas orebody, numerous smaller orebodies, consisting of flat-dipping lenses, extend in all directions, and add considerably to the total tonnage of ore. The area had been extensively worked by former operators, and a maze of old workings was accessible, extending downward to elevation 38.

The principal part of the orebody lay in a position favorable for the glory-hole method. The smaller and outlying extensions of the orebody were mined also by small glory holes from the surface, or by underground open stopes on the veins which were equivalent to glory holes insofar as little or no shoveling was required.

Catillas Orebory

The shape and extent of the Catillas orebody will be apparent upon reference to Figures 1 and 2,B. This orebody had the following principal features:

1. It had been extensively worked by former operators, and the mass of the orebody was badly shattered by the caving of a large proportion of the

old workings. Many of the old stopes were filled with material which would constitute ore under present conditions.

2. The silver content of the orebody was somewhat higher than that of the Hilltop, but not sufficiently so to justify the use of a costly underground mining method.

3. The wall rock adjoining the orebody was relatively hard, and undisturbed by old workings.

4. Certain blocks of very good ore were hard and solid, and would require special attention to insure their clean extraction.

5. The orebody, as indicated in Figures 1 and 2,B, lay in two distinct parts, called the North and South branches, separated by a block of waste.

6. The 0-meter haulage way through which the Hilltop orebody was being mined passed through the upper part of the Catillas orebody. Production from the Catillas orebody, therefore, could not be started until the Hilltop was practically exhausted.

7. This orebody ranked next in size to the Hilltop, and would be required to produce the bulk of the daily tonnage when the latter was exhausted.

Nos. 6 and 7 were the deciding factors in the choice of a mining method. Glory-holing was the only method that would meet the requirements for daily tonnage. The selection of this method was, however, fortified by other considerations. An efficient glory-hole organization was available and the mining of this orebody would follow naturally upon the Hilltop operation. In spite of the waste stripping that would have to be done, experience in the Hilltop operations indicated that the glory-hole method would give a lower mining cost on the Catillas orebody than any other method. The block of waste between the two branches of the orebody presented a problem. If silver prices were high, this block would pay for extraction under a cheap method and dilution from it would not be a vital consideration. If silver prices were low, this block of waste would have to be separated, and this separation would be more easily accomplished under the glory-hole method than under any other. The shape of the orebody was peculiarly favorable to the glory-hole method. Although complete extraction of the orebody by glory-hole mining would not be possible because of the excessive stripping of waste that would be required, the bulk of the ore could be rapidly and cheaply extracted by this method, leaving the lower and richer portions of the orebody in a favorable condition for extraction by other methods, such as top slicing or block caving.

The Catillas and San Nicolas orebodies were mined concurrently, after the exhaustion of the Hilltop orebody.

Pilar Orebody

The position, form, and extent of the Pilar orebody are shown in Figures 1 and 2,C. The orebody had the following outstanding characteristics:

1. It was long, narrow, and deep-seated.

2. Its silver content was nearly twice that of the Hilltop and Catillas orebodies.

3. A high-grade streak in the vein had been mined by former operators, and the old stopes had either caved or were filled with material now classed as ore. The payable ore was made up of these old fills, pillars left in the vein, and very irregular areas of good ore extending into the walls.

The governing factor in selecting a mining method for this orebody was its relatively high silver content, which made it necessary to mine it with a minimum of dilution. Glory holing was rejected because of the prohibitive amount of waste that would have to be stripped. Any overhand stoping method was out of the question, because the badly broken nature of the orebody would make the operation extremely hazardous, if it could be carried on at all. Any caving method appeared certain to result in heavy dilution in the irregular parts of the orebody, and might entail the entire loss of some of these extensions.

Top slicing was selected as being the cheapest method that would permit complete and clean extraction of the ore with minimum dilution. A new organization would have to be trained; but, on the other hand, the daily rate of production from the orebody was not a factor, because of other and ample sources of production.

San Pedro Orebody

The position, form, and extent of the San Pedro orebody are shown in Figures 1 and 2,A. This orebody represents an irregular extension of the Hilltop orebody at the intersection of the San Pedro fault with the main fractures of the Catillas system, in a zone characterized by pronounced transverse fracturing. Both orebody and walls were exceptionally firm and solid, though honey-combed by the open stopes of former operators. The orebody was irregular in form, and consisted of streaks of high-grade ore between masses of lower grade but payable material. While relatively unimportant in comparison with other orebodies, it contained an important tonnage of higher than average grade silver ore.

The orebody could not be completely mined without weakening the walls of the Catillas orebody and thus imperiling the glory-hole operation. If mined by the glory-hole method, as part of the Catillas operation, a heavy tonnage of waste would have to be stripped, which would have to be hoisted at a time when the general shaft was already taxed to capacity by the Catillas operation.

In view of all conditions, the decision was reached to mine out the areas of better-grade ore by open-stope methods, which as finally developed amounted practically to underground glory holing. Ample pillars of lower-grade ore would be left, which could be recovered by caving methods after completion of the Catillas glory-hole operation.

METHODS OF SAMPLING AND ESTIMATION

The tonnage estimate of the Hilltop orebody, made by the Fresnillo Mining Co., was based on extensive sampling of quarry faces and floors, and of a few crosscuts driven by the company. Other parts of the orebody were sampled from the maze of old workings by drilling more than 700 holes with hand steel. These holes usually were 3 meters deep, and were drilled at right angles to the main fractures at any given point. In taking over the property the Mexican Corporation accepted the tonnage as stated, but reduced the estimated silver content by 10 per cent. The operating results when the extraction of the orebody was completed indicated that the true silver content of the orebody was about half-way between the two estimates.

The following general remarks relate to the orebodies other than the Hilltop that were developed by the Mexican Corporation.

The orebodies were developed on the several levels by parallel crosscuts at intervals of from 50 to 100 feet. These crosscuts were carefully channel-sampled in blocks 2 meters long. Assay maps were made for each level, and upon these maps the orebodies were delimited, giving consideration to the mining method to be used. For instance, sharp reentrant angles could be allowed where top slicing was to be used but were not permissible where the ore was to be mined by glory holing because of the difficulty of closely controlling the operation. In thus "smoothing" the outlines of the orebody some ore was excluded and some low-grade material was included at its assay value; the intention was to make an estimate of recoverable tonnage that would be checked by operating results.

Vertical and parallel cross sections at intervals of 5 meters were constructed from the assay plans. At each level an assay value was assigned to each cross section by weighting its relation to the two nearest crosscuts. An assay value for each ore area on the cross section was calculated by weighting the assay value of the upper and lower levels bounding it. Ore areas were then measured by planimeter on each of the cross sections, and a total assay value for each cross section was calculated by weighting its component parts.

Cubic contents were calculated by averaging the areas of each pair of adjacent cross sections and multiplying by the interval of 5 meters between them. The silver content of each of the 5-meter blocks was obtained by weighting the assay values of the two cross sections enclosing it.

The specific gravity of the ore was determined by experiment. The figure used varied from 2.2 to 2.5, depending on the proportion of broken ground, old fill, and solid material.

In estimating silver content, a preliminary discount of 10 per cent was applied to actual assays, as allowance for probable errors in sampling and assaying. As an additional precaution, the work of the local sampling organization was checked by resampling a considerable number of crosscuts under the close and detailed supervision of an outside engineer.

In general the estimates of silver content have checked quite closely with operating results. Recovered tonnages have usually been somewhat higher than the estimates, probably because of the fact that wherever it was difficult to determine specific gravity accurately, because of the presence of old fills, the more conservative figure was used.

In laying out the Catillas glory holes and estimating the ore and waste contained in them, theoretical contour maps were drawn to represent the interior surface of the finished pit. The orebody had its greatest horizontal extent at about the 44-meter level. The first contour drawn was therefore laid out at this elevation to enclose the entire orebody within a smooth outline containing as little waste as possible. From the 44-meter level to the surface, contours were constructed at 5-meter vertical intervals, the slope in each part of the glory hole being governed by the character of the ground. In the parts of the glory hole where the ground was firm and solid a slope of 2 to 1, or 63-1/2° was used. According to experience in mining the Hilltop orebody this was the maximum safe slope for solid ground. Its use in the Catillas orebody, which is now nearly mined out, has been justified by the results. Where the ground was broken or soft a slope of 45° was used. From the 44-meter level downward the governing factor in the shape of the pit was the location of the several chutes and grizzlies at the 105-meter level. Contours from the 44-meter level downward were also constructed at 5-meter vertical intervals, with maximum slopes of 63-1/2° to the bottom of the theoretical pit. These contours were then transferred to the cross sections, and the theoretical slopes were constructed. Each cross section was then studied in detail as to (a) ore left outside the glory hole, and (b) waste-to-ore ratio of the ground at the periphery of the glory hole. In considering the inclusion or exclusion of any given section of ore by modifying the shape of the pit, the amount of waste affected by the change, or the waste-to-ore ratio, was considered in each case, as well as the grade of the ore. The higher the silver content of the ore, the higher would be the economic limit of the waste-to-ore ratio. Ore on the periphery of the pit was considered in detail with this viewpoint, and the final outline was arrived at by the cut-and-try method. In parts of the pit where the orebody was irregular in both plan and section the contours had to be revised numerous times before the final design was determined. Falling silver prices in recent years have made it necessary to revise the pit design several times to meet the new conditions.

Waste tonnage was determined in the same manner as the tonnage of ore.

Figure 3 is a typical cross section of the Catillas orebody, showing the relation of the estimated and actual pit lines to the orebody itself.

METHODS OF DEVELOPMENT AND STOPING

General Development

Hilltop Orebody

The outlines of the Hilltop orebody were closely delimited by the maze of quarries in Proaño Hill, and by crosscuts driven from them at an elevation of 25 meters above the 0 level.

Figure 4 shows the general plan of the haulage way driven at the 0-meter elevation to permit large-scale mining of the deposit by the glory-hole method. The bottom of the orebody was from 20 to 60 feet above the haulage-way, except for the two downward extensions represented by the Catillas and San Pedro orebodies, so that practically all of the haulage way was driven in waste. All of the Hilltop orebody was extracted through this haulage way, as well as a part of the Catillas. Ample room for a mill site remained on the hillside below the portal of the haulage way.

The standard cross section of the haulage way was 10 feet wide by 9 feet 6 inches high. Timber was required in only a few places. All curves were of 100-foot radius. The track was of 50-pound rails laid to a 36-inch gage on 6 by 8 inch by 6-foot pine ties at intervals of 2 feet 6 inches; it was well ballasted. The grade was 0.5 per cent in favor of the load.

Crosscuts were driven to prospect several favorable blocks of ground. These crosscuts were 3 feet 6 inches wide and 6 feet high, and were driven without track. Wheelbarrows were used to handle the material to the main haulage ways, where it was shoveled into cars. This work was done very cheaply.

Glory-hole loading stations were installed at intervals of 150 feet along the main haulage ways, staggered so as to be on the corners of equilateral triangles. Figure 5 illustrates the standard loading station installation.[4] Raises of circular cross section 10 feet in diameter were driven from the grizzly chambers to the surface. Advantage was taken of old workings in planning these raises; they were driven at various slopes and directions to reach their objectives with the least possible amount of work. The raises ranged from 125 to 200 feet in length.

Catillas and Pilar Orebodies

Coincident with preparations for mining the Hilltop orebody, unwatering of the deep parts of the old mine was begun through an old shaft to the east of the Pilar orebody. From this shaft access was had to the Pilar and Catillas areas through an old level at an elevation of from 105 to 112 meters. The old workings were useless for any modern mining purpose, as they were

4 - Baker, T. C., Fresnillo Glory-hole Mining Practice: Eng. and Min. Jour.-Press, vol. 116, Dec. 1, 1923, pp. 931-942.

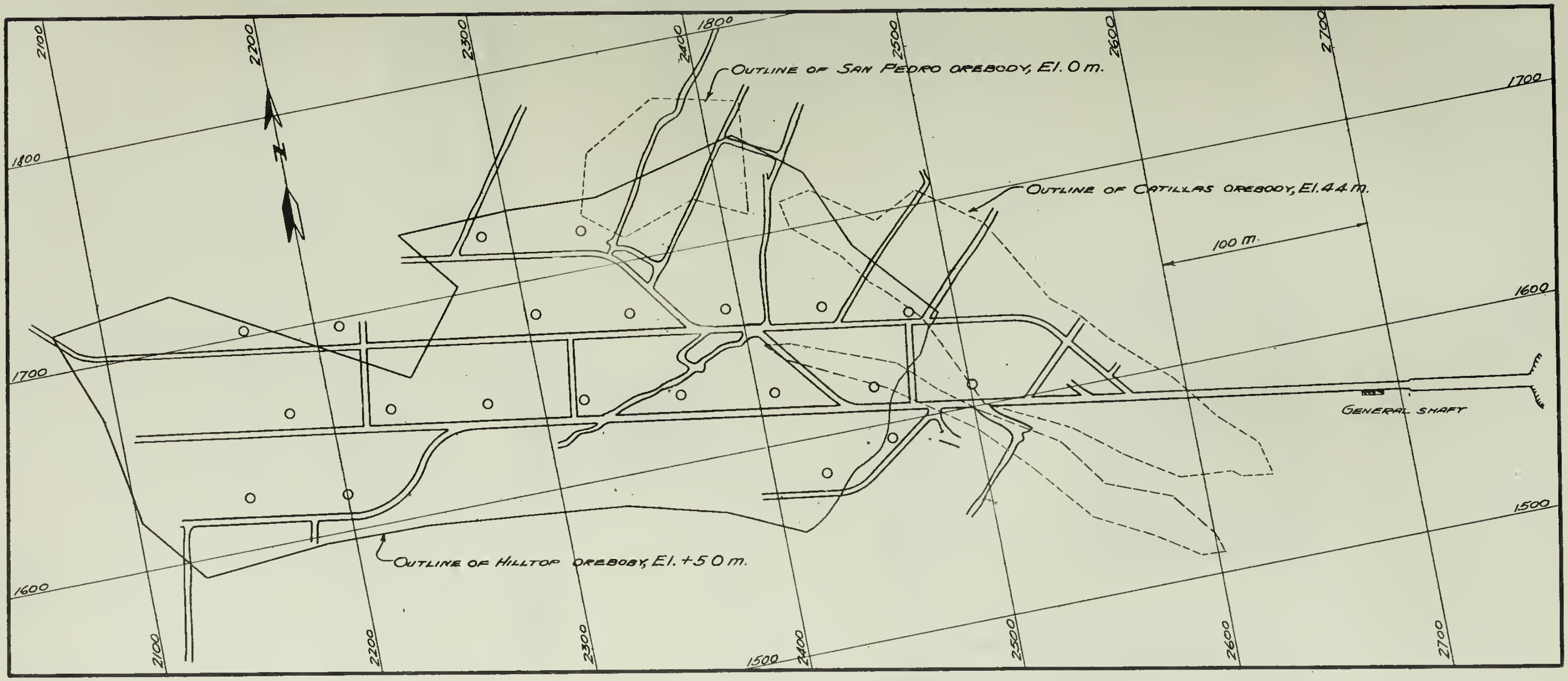
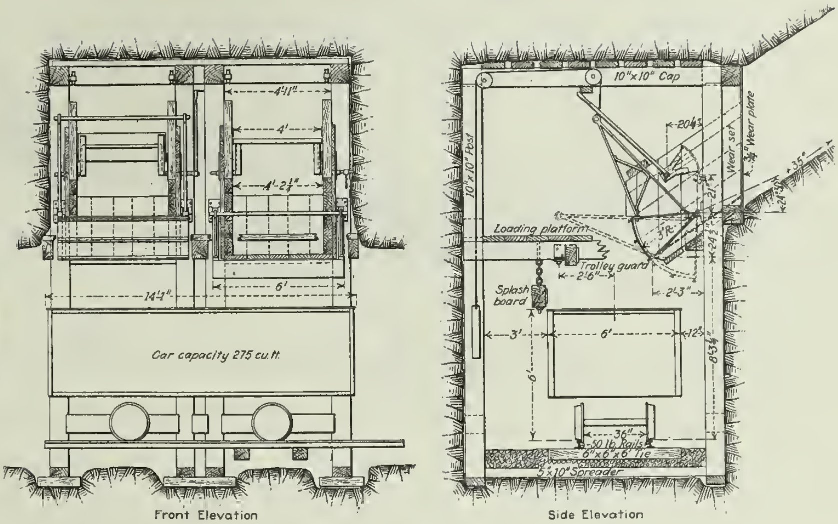
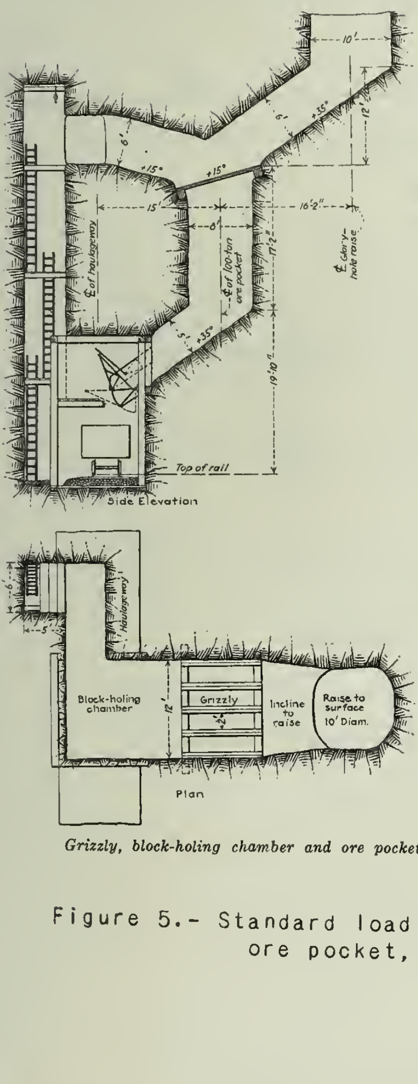


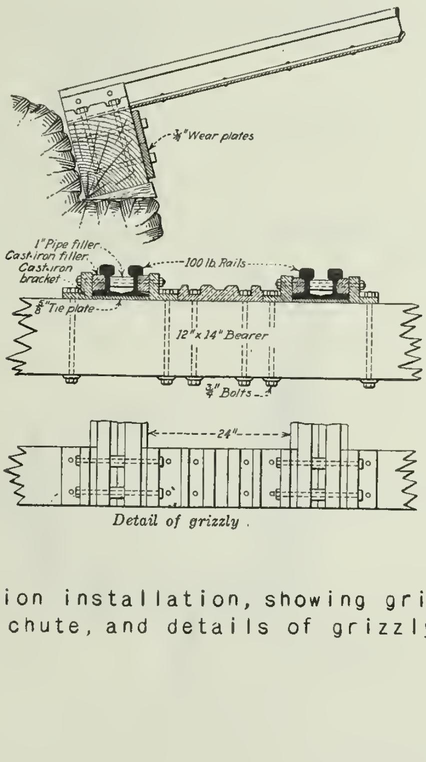
Figure 4.- General plan of haulage way at 0-meter level



Loading chute



Grizzly, block-holing chamber and ore pocket



Detail of grizzly

Figure 5.- Standard loading station installation, showing grizzly chamber, ore pocket, loading chute, and details of grizzly construction

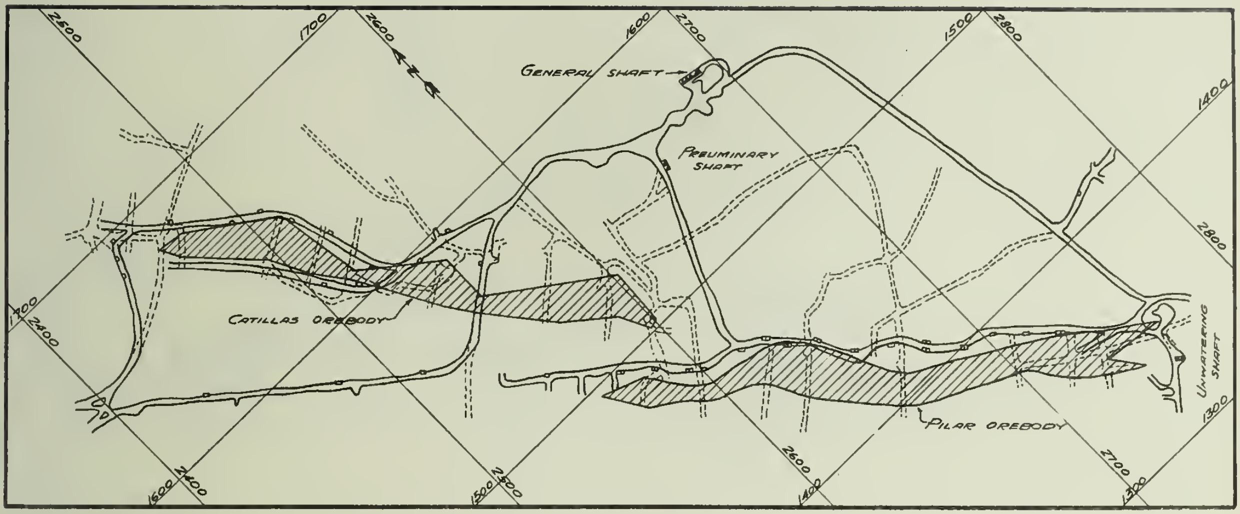


Figure 6.- Part of 105-meter level. Old workings and preliminary crosscuts shown in dotted lines

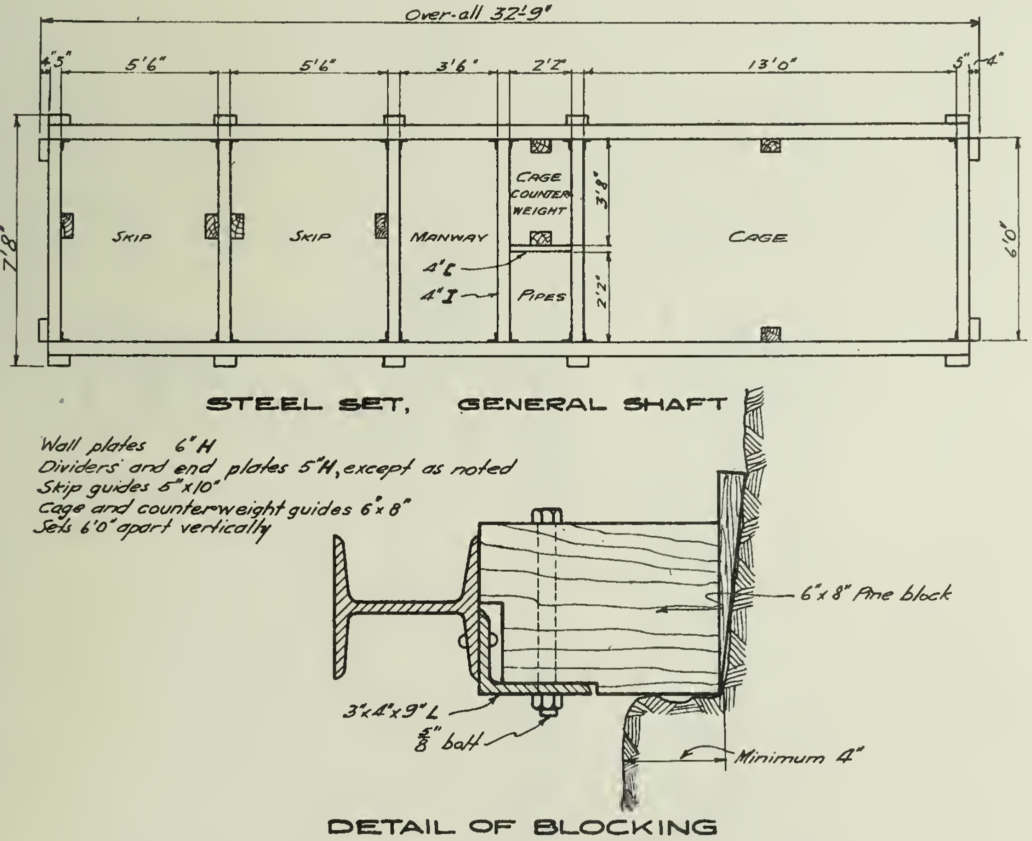


Figure 7.- Steel set and method of blocking, general shaft

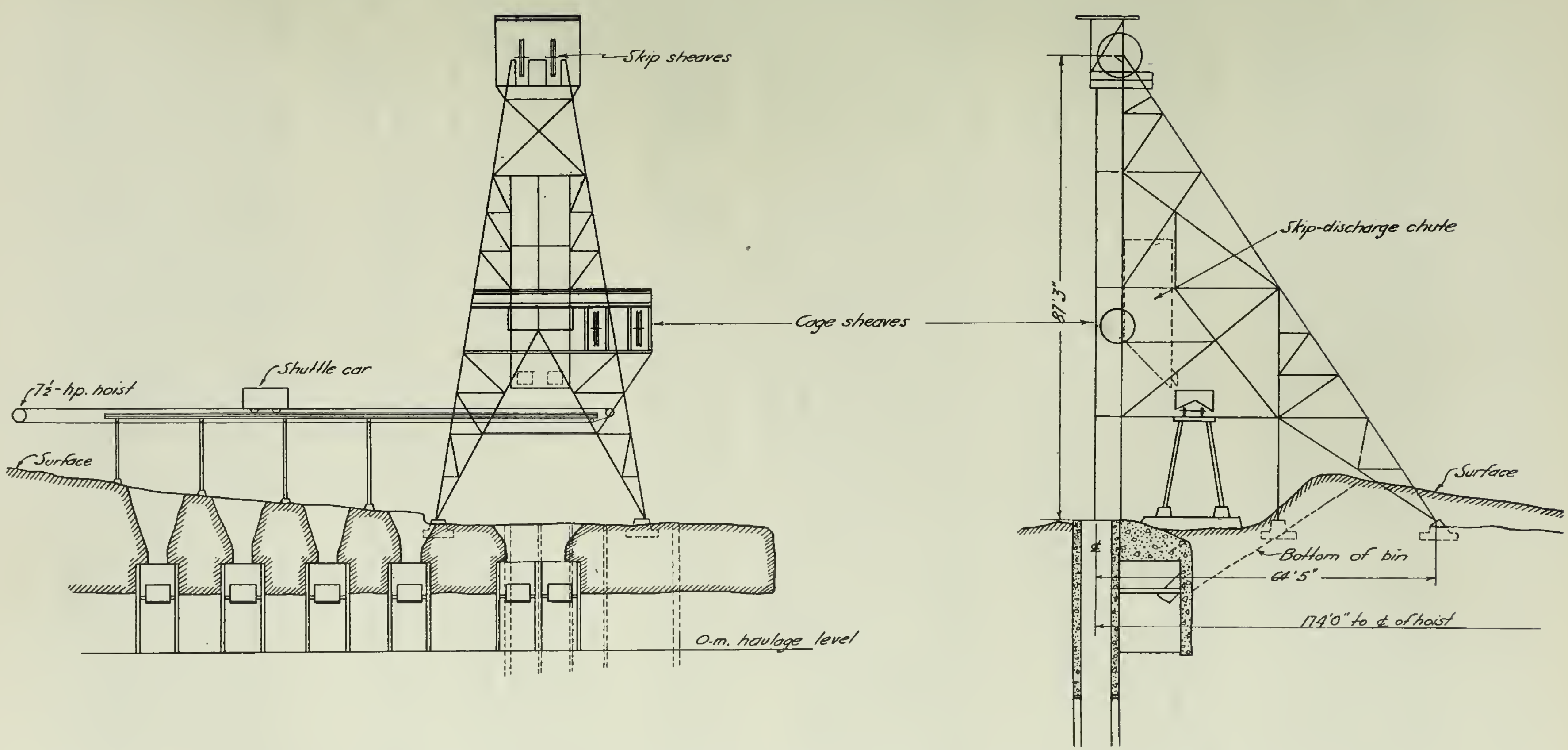


Figure 8.- Front and side elevations at General shaft, showing headframe and arrangement of loading bins

crooked and drifts varied in elevation as much as 10 meters between ends. However, they gave immediate access to the desired parts of the mine, and made it possible to determine the outlines of the orebodies very quickly and cheaply. Three by six foot crosscuts were driven from these old workings at advantageous points, the material from the crosscuts being stored in the old workings. Figure 6 is a plan of the 105-meter level, and shows a part of these old workings and the crosscuts driven from them.

With the outlines of these two orebodies established and a sufficient tonnage of ore indicated, comprehensive development was undertaken. An old shaft, extending to the 105-meter level and conveniently located, was equipped for temporary service, and levels were driven from it at elevations of 44 and 105 meters. These two levels were placed to take advantage of certain old drifts that were in good condition and at suitable elevations to be used. In consequence, these levels were opened up very rapidly.

The 44-meter level was driven primarily for the purpose of development rather than extraction, and the Catillas and Pilar orebodies were thoroughly explored at this horizon. Drifts and crosscuts were 5 by 7 feet in cross section. In and near the orebodies the workings had to be timbered. Subsequently a part of the level was enlarged for haulage to 7 feet 4 inches high above the rail, 5 feet 6 inches wide at the top, and 7 feet 4 inches wide at the bottom inside of the timbers. Caps were of 10 by 10 inch timber and 7 feet long. Posts were of 10-inch round timber 8 feet long.

The 105-meter level, part of which is shown in Figure 6, was established as the principal haulage level, and the main drifts were placed to skirt the orebodies. These workings were of the standard haulage drift cross section described above.

As unwatering of the old mine continued, access was gained to the Pilar area through old levels at elevations 142 and 165, and additional crosscutting was done from these old workings. The Pilar orebody was proved to extend downward to within a few meters of elevation 165, and the lower limit of the Catillas orebody was determined to be at approximately elevation 135.

A haulage level was driven from the General shaft at elevation 165. This level had to be timbered in and near the orebodies, and was driven with the standard cross section.

General Shaft.--The General shaft was sunk for mining the Catillas and Pilar orebodies. It was located alongside of the 0-meter haulage way so that it would be served by existing tracks. The collar of the shaft was placed at a suitable elevation to provide ample storage in the surface bins above the 0-meter haulage way at the minimum cost. The shaft is 7 feet 8 inches wide by 32 feet 9 inches long and is lined with steel sets; a horizontal cross section of the shaft and a detail of the method of blocking the steel sets are shown in Figure 7. The general surface arrangement of the shaft and ore bins is shown in Figure 8.

The shaft was sunk from the surface in full cross section. At the same time raises of small cross section were driven from the 44-meter and 105-meter levels. After the raises were connected this section of the shaft was stripped and the steel placed from the top downward, the waste from stripping being drawn off through the raises. The method was cheap and safe. Below the 105-meter level the shaft was sunk and lined in full section.

The shaft is in very good ground and is largely without lagging. Gunite was used in some places where the ground had a slight tendency to slack and scale. Masonry was used to support a number of large slabs that developed after shaft timbering was complete. In several places where lagging was required, worn rods from the Marcy mills were used. A hook was forged on one end of the rod to fit over the outside flange of the steel wall plate. The rods hang vertically and the lower ends engage the next set below. They are spaced from 10 to 20 inches apart, depending on the thickness of the rod and the weight of to be supported. Sheets of corrugated iron were placed behind the rods with the corrugations horizontal, and the space between the sheets and the ground was tightly filled with waste rock. The corrugated sheets were painted on the back side with heavy crude oil to retard rust. Installations of this type made in 1923 have required no subsequent attention.

Ladders are of steel and manway landings are of concrete. The division between the manway and skip compartments, above the 135-meter level pockets, is of 3/4-inch mesh steel screen. Below these pockets, the division is of 2 by 10 inch planks, fitted tightly in the channels of the 5-inch H-beam dividers. A tight division is necessary to confine the loading spill to the skip compartments. This timbering is the only fire hazard in the shaft, but no satisfactory substitute for it has been found within reasonable cost limits. The wooden lagging has the additional great advantage that it can be replaced safely from the manway compartment without stopping the skips. As the shaft upcasts warm, moist air, this lagging has been damp at all times, and represents only a small fire risk. With the exception of that at the 105-meter level, all the shaft stations are in hard ground and are untimbered.

The shaft is concreted outside of the steel sets for 30 feet below the collar. The headframe is of steel. Ample storage capacity is provided in the bins at the collar, at a very moderate cost.

The cost per foot of excavating and "timbering" the General shaft from the collar to a depth of 800 feet was as follows:

Cost per foot (U. S. currency)

Excavation	$41.94	
Timbering (steel sets)	116.61	
Tramming and hoisting	11.10	
Explosives	9.50	
Miscellaneous	14.93	$194.08
Labor	90.18	
Supplies	99.02	
Power	4.88	$194.08

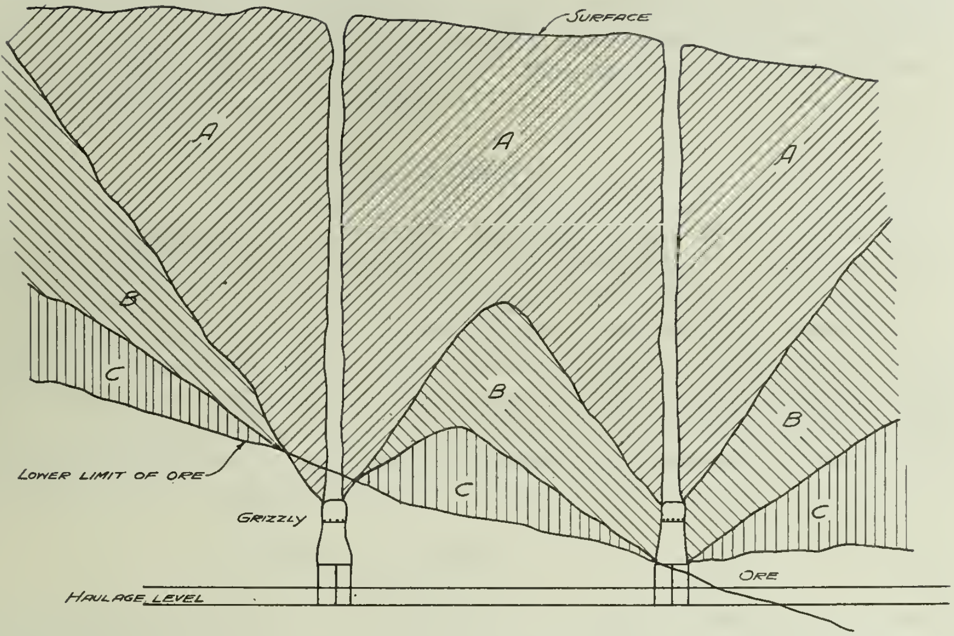


Figure 9.- Typical section of glory hole: A, First stage; B, second stage; C, final stage

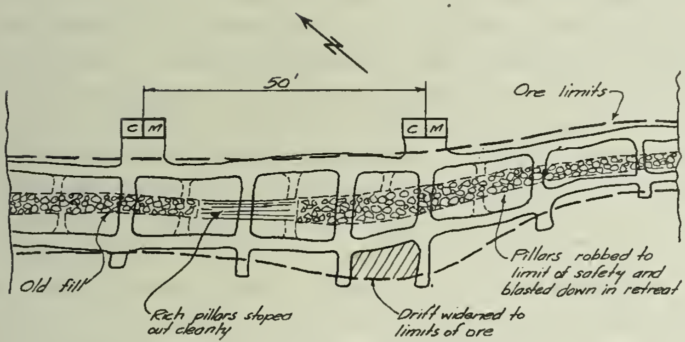


Figure 10.- Typical caving sublevel, Pilar orebody, elevations 30, 37, 44, 54

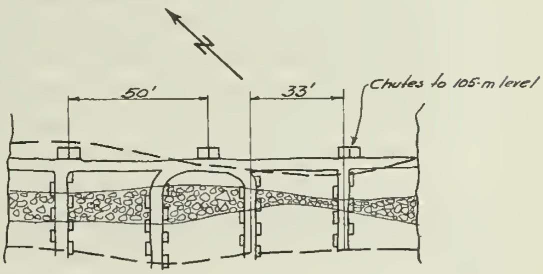


Figure 11.- Typical hand-tramming caving level, 65-meter level, Pilar orebody. Crosscuts solidly timbered, with drawing chutes on both sides in alternate sets

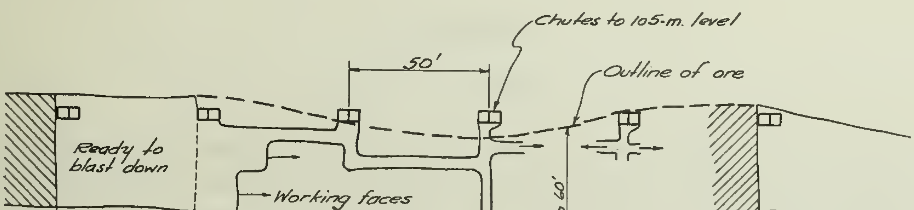


Figure 12.- Typical top-slice floor

The above figures include all items chargeable against the shaft, calculated as cost per vertical foot of finished shaft. The cost of the concrete collar, the loading pockets for the 105-meter and 165-meter levels, and the construction of the surface bins, is included. The cost of the headframe, hoist, and other equipment is not included.

San Nicolas Orebody

This orebody was developed by sublevels at elevations 0 and plus 15. Ore and waste from development were stored in old workings. A drift on the 105-meter level was extended to a point underneath the orebody, and an ore pass was driven which connected to old workings at elevation 38, where a grizzly was installed. Branches from this ore pass above elevation 38 reached the most important parts of the ore body so that the ore, when broken, ran direct to the grizzly without shoveling or tramming. Ore from parts of the orebody beyond the economic reach of branch raises was trammed on the 0-meter level to one of the branch raises.

Development Details

The adoption and enforcement of standard drilling and blasting practice has not been practicable at Fresnillo, principally because of widely varying ground conditions. Rock drills have been extensively tested and the most efficient models have been supplied. Much drifting and crosscutting have been done with unmounted jackhammer drills weighing 55 pounds; mounted machines have been used where the ground was harder or greater speed was desired, especially in the larger heading. Raising has been done with stoper drills, and in some instances with mounted machines. Drill bits are of the standard double-taper pattern. Three kinds of hollow drill steel are used: 7/8-inch hexagonal for the jackhammers, 1-inch round for the stopers, and 1-1/4-inch round for the mounted machines. The first mentioned has been used in the glory holes; in development headings jackhammers have been equipped to use the 1-inch round steel. All drilling in development headings is done with wet drills. Either 30 or 40 per cent strength gelatin dynamite is used in development work. Mucking is done by hand, using a D-handled, round-pointed shovel.

Stoping

Hilltop Orebody

Glory-Hole Method.--Figure 9 is a typical cross section through the Hilltop glory holes. The slope in the glory holes was maintained at 55° or steeper as long as possible, or until the collar of the glory hole had been mined down either to the bottom of the orebody or as close to the grizzly as possible and a flatter slope became imperative. When the slope is steep the ore is more thoroughly broken up by falling after the blast and grizzly expense is greatly reduced. This effect is most notable, of course, when working high up on the sides of the slope. The first cut in each cycle was taken at the bottom of the glory hole around the collar of the raise. Successive benches 8 to 10 feet in height were then mined; the direction of retreat was upward toward the rim. When working in loose ground this procedure was occasionally reversed from

considerations of safety. When the bench had been completed to the rim, all sides of the glory hole were carefully barred down while the glory-hole raise was being emptied, so that the collar of the raise could safely be reached to start a new cycle. It was advantageous to maintain the steep slope as long as possible, to avoid the necessity of working the broken ore into the glory hole by hand. In cases where the bottom of the orebody was at a distance above the grizzly it was profitable to start the cyle by taking a cut in waste around the collar of the glory hole. This waste was handled as such and not mixed with the ore, and the procedure was followed until the cost of breaking and handling the waste overbalanced the cost of mucking in the glory holes. Thereafter, slopes in the glory holes were successively flatter with each cycle, until a minimum of approximately 35° was reached. With slopes flatter than 35° it became impossible to mill down the ore without hand mucking. To extract the ore remaining between glory-hole raises, a horizontal bench was started at the bottom of the ore and wheelbarrows were used. Although scrapers have not been used in the glory holes at Fresnillo, the writer believes that under favorable conditions, and particularly in irregular orebodies, large scrapers could be used very successfully with glory-hole mining. Raising would be reduced and the mining of some waste to create safe slopes might be avoided.

Where the ore extended below the grizzly the grizzly was finally removed and the glory-hole cycles were started at successively lower levels--in some cases until the chute mouth was reached. During this stage as much block holing as possible was done in the glory holes to avoid excessive blasting in the chute mouths with consequent loss of time in loading and tramming. Production from the glory hole was relatively small during this final stage, but this was no great disadvantage, as the loss of production was easily made up from other glory holes. In a few places the tonnage remaining between glory holes was sufficient to justify the installation of intermediate loading stations and short raises without grizzlies.

Waste stripping in the Hilltop orebody was small in amount, and consisted mainly of the separation of blocks of waste surrounded by ore, plus a small amount of material from the rims of the holes that had to be mined as a safety precaution.

The safe limit of glory-hole slope is a factor that varies greatly in different localities. At Fresnillo slopes in excess of 70° have been carried with entire safety, where they did not exceed 100 feet in height. The steeper slopes were inspected carefully every day. Sloughing occasionally took place, but it was small in amount and always gave ample warning. In planning glory holes where waste stripping was a factor and the ground solid, a slope of 63-1/2° was used; where the wall rock was less favorable, slopes laid out as flat as 45° were used.

Drilling in the glory holes was done with 55-pound jackhammers, using 7/8-inch hollow-hexagonal steel, the light weight of which was an advantage because of the distance it had to be carried by hand. The machines were fitted with powerful blowing attachments for cleaning out the holes. The

pressure at the drills was a minimum of 75 pounds. Drilling was done without water. Water lines in the glory holes would have been an extra item of cost, and upkeep on the dry machines was less than if water had been used. Wet drilling would have made the slopes slippery and dangerous in many places, and it was considered that the dust had no harmful physical effect inasmuch as drilling was done in the open air.

Holes averaged about 9 feet in depth, and were given a burden of between 3 and 4 feet. The average machine performance was about 120 feet of hole per 8-hour shift. All drilling was done on contract on a basis of footage drilled, the rate per foot ranging from $0.028 to $0.036, U. S. currency. The drill crew consisted of a driller, a helper, and a tool nipper, the latter serving two machines. A contractor usually had two machines, or more, and operated one of them himself. Though paid on the basis of footage drilled, he was required to bar down the benches, except in special cases, and to carry his own steel. Foot trails were maintained to all working faces, and steel was carried by the tool nipper from a central magazine on the rim of the glory holes to the machine. All equipment and steel were carried out of the glory holes at the end of the shift and checked in at the magazine to prevent loss. Drillers worked both day and night shifts, the work at night being confined to places of lesser hazard. The average production per machine-shift was over 130 tons for the whole Hilltop operation.

The drilling practice described above was suited to the nature of the ground and the quality of labor available, and it is thought to have given about the best results possible under such conditions. While it was satisfactory in operation and gave excellent results, the contract system used had a serious defect: the contractor was interested only in drilling the hole as quickly and as easily as possible, and its location was a matter of no interest to him. Unless supervision was close and careful, many holes of too great or too little burden would be drilled. At Fresnillo, a "hard-boiled" policy of refusing payment for such holes was adopted, but the point was never reached where the tendency was entirely corrected.

Blasting was done in daylight only, by a special crew of men under a boss of long experience and proved careful habits. The men worked on company account and were obliged to adjust their hours to the demands of the work at different times. Holes were sometimes fired at noon, sometimes at the end of the day shift, and sometimes early in the morning before the day shift came on. Usually, the blasters came on with the day shift and made up the estimated number of primers for the day's work. The primer consisted of a No. 8 cap, one stick of explosive, and a fuse from 6 to 9 feet long. During the day shift, holes were loaded in places where drilling was not going on, and were occasionally blasted at noon. Loading of the holes drilled on day shift began about the time the drillers were finishing their work, and the shots were fired about 40 minutes after the drillers had gone. The explosive used was bulky, with a comparatively low velocity of detonation, and was rated at about 35 per cent strength. It was selected after extensive tests of a number of brands on the basis of tons of ore broken per pound of explosive.

The average charge was about six sticks of explosive to the hole. The primer was placed near the middle of the charge, which was well tamped; the top of the hole was filled with the dry drill cuttings for stemming. The ends of the fuses were assembled in groups of three or four, as far as possible, and the firing was all done by hand, using a short piece of nicked fuse as a spitter. An air whistle was used to give warning of the blast. The work of spitting the holes was divided carefully among the men, each of whom would spit his allotted holes and retreat to the rim of the glory holes over an appointed route. There was never an accident to a blaster in this kind of work.

A grizzly opening of 24 inches was used throughout in mining the Hilltop orebody. This wide opening was made possibly by use of large chute doors and 10-ton tipple-dump ore cars on the 0-meter haulage level which delivered directly to the gyratory crusher at the mill.

When working high up in the glory holes, and while the slopes were steep, the ore was well broken up by falling, and the amount of block holing required on the grizzlies was comparatively small. Grizzly expense increased very materially when working around the collars of the glory holes and when the slopes were flat. Grizzly men worked in pairs, each pair tending from one to three grizzlies, depending on the coarseness of the ore and the demands of production. Point bars and 8-pound hammers were used to work the ore through the grizzly until it became clogged with large rocks, which would then be drilled and blasted. A 21-pound jackhammer was employed for blockholing, using 7/8-inch hexagonal steel; drilling was without water. Most of the holes drilled were less than 12 inches deep. The men retreated to the haulage level below when the shots were fired. Both the bulk explosive previously mentioned and 30 per cent strength gelatin dynamite were used in this work. Only 4 per cent of the total explosive consumed in mining this orebody was charged to secondary blasting on the grizzlies.

The glory-hole raises had an original diameter of 10 feet, and soon wore out much larger. They never "hung-up" except in raily weather, and not then if worked continuously so that the ore was kept moving and not permitted to pack.

The brow of rock over the grizzly on the side next to the glory hole was found to wear severely, so that eventually the grizzly would become completely flooded with ore and be difficult to work. It was necessary in such cases to install a heavy baffle to hold back the ore in the glory-hole raise. Steel beams were used for baffles in some instances, in others the baffle was made of four 12 by 12 inch timbers, reinforced on the wear side with 1/2-inch steel plate and faced with 50-pound rail riveted to the plate in a vertical position. The baffles usually were set in hitches cut in the rock; in some cases where the raise had worn wide as well as high it was necessary to line the sides of the hole with masonry to reduce the opening to its original width so that the baffle would not have too great a span.

Under favorable conditions, 1,650 tons of ore has been put through a single grizzly in an 8-hour shift. In excess of 22,000 tons of ore has been passed through a single grizzly in a month, working three shifts a day. It was never possible to work a grizzly at full capacity for an entire month because it was never desirable to concentrate production in a single small area for a period of that length. In the Hilltop orebody the absolute capacity of a glory hole, when slopes were steep and conditions favorable, would have been in the neighborhood of 35,000 tons per month. In actual practice a monthly production of 80,000 tons was spread over about 10 glory holes in order to maintain a silver content in the ore produced that would correspond to the average grade of the orebody. The average rate of production in the Hilltop ore body was between 8,000 and 10,000 tons per grizzly per month.

The effect of rainfall has to be considered in planning any open-cut mining operation. Rainfall at Fresnillo, though slight, has been of considerable importance at times. It amounts to from 10 to 12 inches annually, and occurs as heavy showers of short duration, concentrated in the last six months of the year. The time lost by ore-breaking crews is negligible, but during rainy weather work in the pits is confined as much as possible to the upper parts, as the moisture tends to loosen masses of rock which are normally unstable. The most serious effects of rain on the mining operations were the tendency of wet ore to pack in the raises, and the flooding of underground workings with mud and water from the pit drainage. The former difficulty was met by keeping the ore moving in the raises, or by drawing them nearly empty, as stated above. This sometimes necessitated blasting twice a day or oftener. The flooding of the tracks on the haulage often stopped production for short periods. The water could be drained off into lower workings, but the mud had to be shoveled into cars. However, tramming on a reduced scale always could be resumed after a few hours, and only rarely was more than 24 hours needed to complete the clean-up.

The most serious effect of rainfall on the Fresnillo operation as a whole was at the mill, where the capacity of the crushing plant sometimes fell off as much as 60 per cent when the ore was very wet.

Catillas Orebody

Glory-Hole Method.--Glory-hole loading stations for the Catillas orebody were installed on both the 44 and 105 meter levels, at irregular intervals, the irregularity being due to the broken character of the orebody and the consequent necessity of selecting favorable places even at the sacrifice of the desired interval. The standard loading station set is an exact replica on a smaller scale of the set used on the 0-meter level, except that single instead of double chutes were used.

Figure 3 is a typical cross section of the orebody, showing the ore outlines, estimated and actual outlines of the glory hole, and the arrangement of the glory-hole raises. A string of raises was put up from the 44-meter level along the north side of the orebody to facilitate separation of

the V-shaped body of waste that lay partially above the ore. These raises were all provided with grizzlies about 30 feet above the 44-meter level, similar in design to those used in the Hilltop orebody. Long raises were driven from the 105 to the 44 meter level, taking advantage of old workings wherever possible, and continued through to the surface. Grizzlies were first installed about 30 feet above the 44-meter level, and as the operation progressed, were moved downward and installed successively at the 44, 75, and 95 meter elevations. Because of the sticky nature of some of the ore it was impossible to operate with high raises above the grizzlies, as such raises constantly hung up and gave trouble. The space in the raise below the grizzly was seldom filled, and never in rainy weather; consequently, this part of the raise never hung up.

The method of installing the grizzlies varied much in this orebody. Above the 44-meter level several of the raises to the surface required solid cribbing, and grizzly stations were constructed of heavy timber adequately protected with steel wear plates. In some of the raises masonry ore pockets 10 feet square inside and 30 feet high were built up from the 105-meter level to carry the grizzlies at elevation 95. However, all of the principal features of the grizzly design used in the Hilltop orebody were retained in the Catillas operation, as experience had proved that they gave the most satisfactory operating results.

The primary drilling and blasting practice followed substantially that developed in the Hilltop operation. Because of the broken condition of the orebody, the footage of holes drilled per machine-shift was notably less than in the Hilltop orebody, and the labor of barring down was correspondingly greater. Accurate figures as to tons per machine-shift are not available, but probably would be from 10 to 15 per cent under the Hilltop figure, which was about 130 tons. The favorable shape of the orebody, as seen in Figure 3, allowed steeper slopes to be carried throughout the operation. The orebody has been mined down practically to the 85-meter elevation, with only an insignificant amount of hand mucking. Greater precautions for safety had to be taken than in the Hilltop operation, and the slopes were constantly and carefully inspected for loose rocks and cracks. The glory-hole operation has been continued until the walls of the pit in the highest part at the northwest end are about 450 feet high, vertically, and are standing at an average slope of more than 60°. There has never been a serious accident due to a fall of rock from the walls, and only a few accidents that occasioned a loss of time for the injured men.

A grizzly opening of 16 inches was adopted for the Catillas orebody. A smaller opening was made necessary by smaller tramming equipment and smaller chute gates on the 105-meter level, and by the fact that the ore had to pass through loading hoppers and be hoisted in 5-ton skips in the General shaft. Because of the smaller grizzly opening, secondary blasting was a somewhat more costly operation than it had been in the Hilltop operation. Production per man-shift was lower, powder consumption on the grizzlies was higher, and the items of grizzly construction and maintenance were more costly. The foregoing were the principal factors accounting for the increased cost of orebreaking in the Catillas operation as compared with the Hilltop.

Pilar Orebody

Former operators had stoped a narrow high-grade streak in the Pilar orebody continuously through to the surface, and these old stopes for the most part had been left filled. From the surface to a short distance above the 44-meter level the orebody was too low-grade to mine. Careful consideration was given to the breaking down of this overburden and to the formation of a cushion of broken ground under which top slicing could be started with safety.

Caving Methods.--Exploration of the upper parts of the orebody indicated that its extent and silver content would not justify top slicing above elevation 89. Caving methods were therefore adopted to recover the ore above this elevation, with the further idea that this operation would adequately prepare the way for top slicing below the 89-meter level by thoroughly breaking the overburden and leaving it in a condition to follow the mat down. At the 44-meter level the orebody was 15 feet wide. Downward the width increased to 35 feet at the 65-meter level. Because of this narrow width, caving levels were driven at short vertical intervals. There were four caving levels, at elevations 30, 37, 44, and 54. (See fig. 10.)

Figure 10 illustrates the procedure followed on these four upper intermediate levels. A drift was driven in the comparatively solid north wall of the orebody, from which crosscuts were driven through the old fill and connected to each other on the south side at the edge of the ore. Where they penetrated old fill, the workings required timbering, and at times had to be spiled. Most of the work, however, was safely accomplished without timber. Old pillars of rich ore in the vein where found were mined out as cleanly as possible. The pillars between the crosscuts were then robbed to the limit of safety and blasted down in retreat. Sometimes the backs of the undercut openings were drilled and blasted, but not as a rule, for the ground usually caved readily when the supporting pillars were blasted out. The ore broke up well, only occasional large blocks requiring blasting in the chute mouths. The work on these sublevels resulted in the clean recovery of about 15 per cent of the better part of the ore, and left the remainder in a condition much resembling a completed shrinkage stope approximately 80 feet high and 300 feet long. The walls of the orebody, while not notably strong, were sufficiently so to prevent excessive dilution while the broken ore was being drawn.

The orebody was narrow, and would justify only one haulage drift and one line of raises from the 105-meter level. The drift was driven to skirt the north edge of the orebody as nearly as possible, and 2-compartment, timbered raises to the 65-meter level were located along it at intervals of 50 feet.

The 65-meter level was used as a hand-tramming level for the caving operation, and had its own independent system of raises. Figure 11 illustrates the method of caving and drawing off the ore at the 65-meter level, which followed in a general way the method then in use by the Miami Copper Co. Chutes opposite each other in contiguous crosscuts were connected by 30° inclined raises driven in the comparatively solid ground outside of the old

fill. These raises were then widened to the limit of safety on the side away from the fill. Pillars between the raises and the old fill were next robbed as far as safety would permit and then blasted down. Drawing from the chutes was then started and carried on as evenly as possible over the entire length of the orebody, so as to prevent funneling. Very little timber was used in the undercutting operation.

Subsidence of the surface followed within a very few weeks after drawing off at the 65-meter level was started, first in a narrow zone closely following the old filled stopes, but soon extending laterally as the walls surrounding the caved ore collapsed. The tonnage and silver content of the ore drawn at the 65-meter level closely approximated expectations. As the cost per ton of ore drawn was quite low the success of the caving operation was indicated.

Following this drawing operation the orebody was undercut and caved at the 79-meter level by methods very similar to those used on the upper sublevels. However, the greater width and higher silver content of the orebody at this elevation required that the operation be carried out more carefully in detail, and care was taken to reach the limits of the ore in all directions, and to undercut it thoroughly. No ore was drawn off at this level.

At the 89-meter level, a complete floor was taken out of the orebody by the square-set method, and again care was taken to reach the ore limits in all directions. The square-sets were blasted down in blocks as large as possible, to get the maximum shattering effect on the ore above.

At the 91-meter level, a combination of top slicing and brow caving methods was used, the details of which are not of special interest. A complete floor was taken off by this method, and all of the broken ore below the 65-meter level was drawn off. A mat of double 3 by 12 inch plank was laid.

Top Slicing.--Top slicing started at elevation 94, and was carried down to elevation 162, at a point 10 feet above the 165-meter level, where the orebody terminated. The method used varied greatly in detail as new ideas were developed from time to time. Figure 12, however, illustrates the general practice. Figure 13 shows the usual method of timbering.

The work was done on company account at the start, but was later put on a contract basis. A unique feature was the system of payment. A prime necessity when using the top-slice method is to get the floor laid as soon as possible. To insure carefulness in this respect, the contract price for ore breaking was fixed at so much for each floor board of standard size that was laid. This price was carefully worked out and was never changed throughout the whole operation. Other payments were made for spiling, which was occasionally necessary, reinforcing of timber, etc., but more than 90 per cent of the money paid the contractor was calculated on the basis of the floor boards laid. Even the driving of advance drifts was put on this basis. After the adoption of this system, cases were very rare where the slice caved before the floor was laid. Moreover, it was to the contractor's pecuniary advantage to lay the floor as closely and tightly as possible.

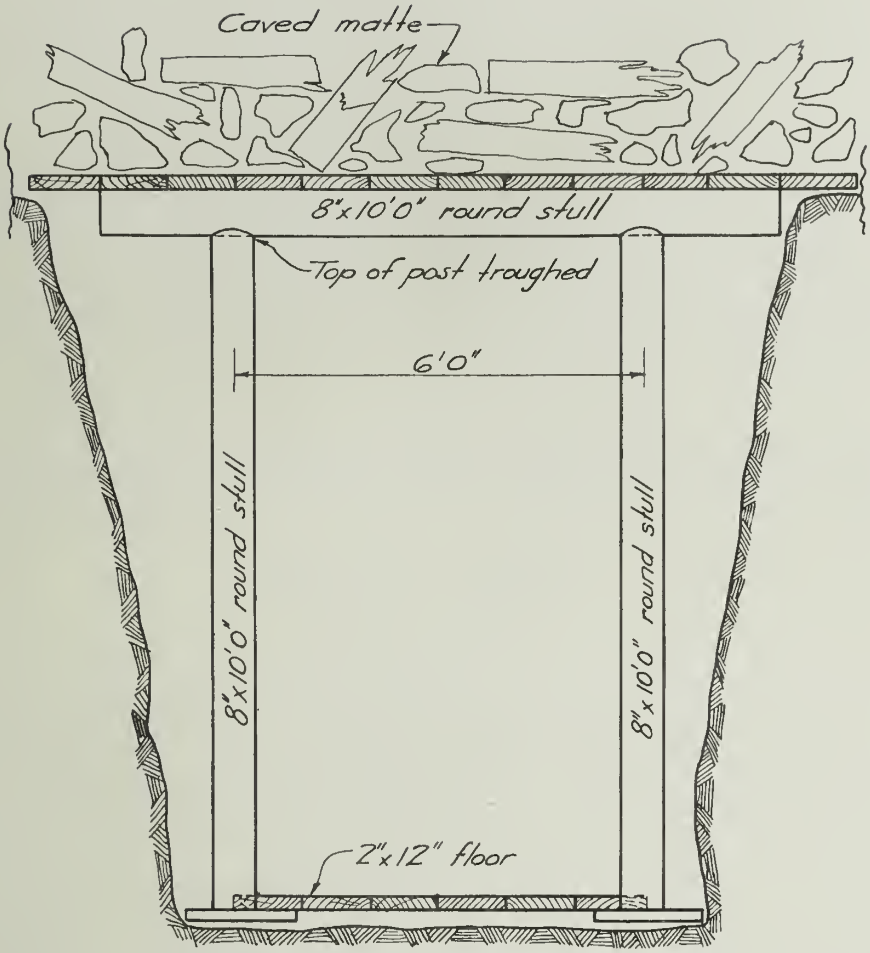


Figure 13.- Typical top-slice timbering, used as shown in advance, at intervals of 5 feet. Extended laterally in stopes

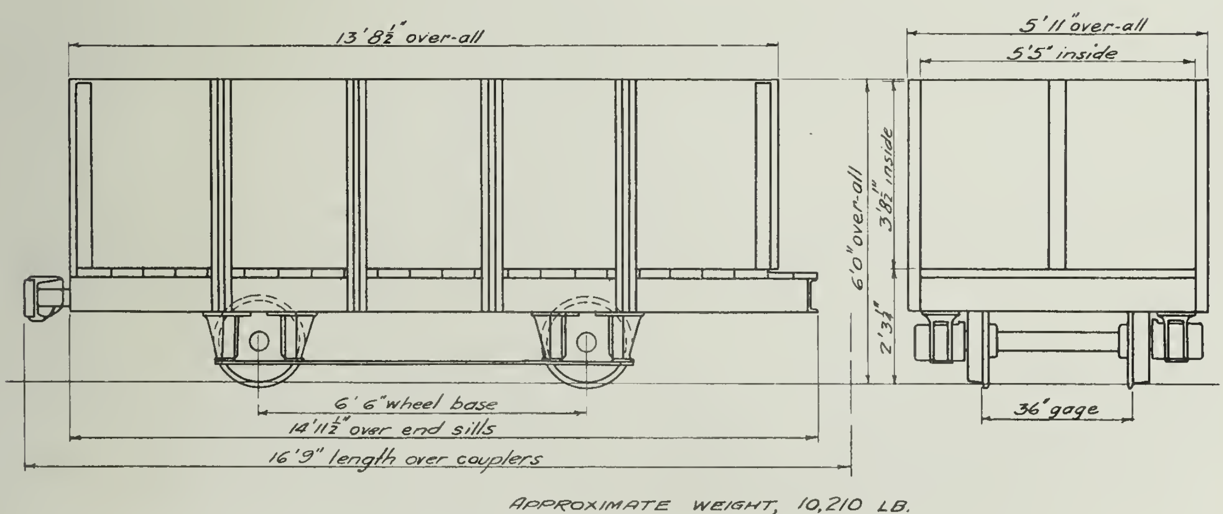


Figure 14.- Mine car

San Pedro Orebody

Open Stopes.--Mining in the San Pedro orebody was confined to small operations that could be carried on without weakening the ground to the extent of jeopardizing the Catillas glory-hole operation. The stopes may be described as small underground glory holes, so planned as to reduce shoveling and tramming to the minimum. Adequate pillars were left to support the ground. The western and northwestern extremities of the orebody were caved and drawn to completion after the limit of extraction by open-stope methods had been reached.

More than 70 per cent of the total ore in the orebody has been recovered, leaving the balance to be recovered by caving methods after completion of the Catillas glory-hole operation.

UNDERGROUND HAULAGE

0-meter Level

The locomotives on the 0-meter level are of the trolley type, weigh 8 tons, and use 250-volt current. Ten cars of the type shown in Figure 14 made up the usual train. The average length of haul for the Hilltop orebody was approximately 1,800 feet. This equipment is now used for hauling ore from the General shaft to the crushing plant, a distance of 500 feet.

Both cars and locomotives have given excellent service as a result of adequate strength of design and a high standard of maintenance. After 10 years of service, during which time 3 locomotives and 22 cars have handled 11,100,000 tons of ore, cars and locomotives are as good as new from the standpoint of the service they give.

For transporting waste to the dump, a 4-yard side-dump car of the Western wheeled scraper type was used with excellent results. The waste dump was managed according to the general practice of large steam-shovel operations.

44 and 105 meter Levels

Tracks on the 44 and 105 meter levels are of 25-pound rail laid to 20-inch gage on 5 by 6 inch by 3 foot 6 inch pine ties spaced at intervals of 2 feet. Grades are 0.5 per cent in favor of the load.

Locomotives are similar to those used on the 0-meter level, and likewise use 250-volt current, but weigh only 4.5 tons. Cars are of the gable-bottomed type with a capacity of 70 cubic feet and a maximum door opening of 1 foot 6 inches. From 8 to 14 cars are handled in a train, depending on the length of the haul and the switching facilities at the different chutes. The average length of haul for the Catillas orebody was approximately 650 feet.

When full production was first reached in the Catillas orebody, two trains were operated on the 44-meter level and one on the 105-meter level. At a

later stage one of the trains from the 44-meter level was moved to the 105-meter level and the remaining one was used principally for handling waste from the north side of the Catillas orebody. In the final stage, after the stripping of waste was nearly completed, all of the equipment was concentrated on the 105-meter level but only two trains were operated. The haulage was much more efficient when done on two levels, because of less interference of the trains. When operating at the 105-meter level, on a three shift per 24 hour basis, in dry weather and when loading conditions are otherwise fair, two trains of from 10 to 12 cars each have a demonstrated average capacity of more than 3,500 tons per day.

165-meter Level

Track specifications were the same on the 165 as on the 105 meter level. Power was supplied by a 4-ton, storage-battery locomotive. The batter was of the Edison alkaline type, had 84 cells, and a deliverable current capacity of about 300 ampere hours. A charging station was maintained on the level. Tramming was on a 2-shift basis, and two batteries were maintained for the locomotive. Cars were of the rocker-dump design, with a capacity of 35 cubic feet. From 8 to 12 cars were hauled in a train.

HOISTING

Ore is hoisted in the General shaft. The skips are of 135-cubic foot capacity, equivalent to 5 metric tons. They have cast-steel crossheads with single safety dogs of special design, and weigh about 4 tons. Sheave wheels are of the bicycle type, 8 feet in diameter. Cables are 1-1/8-inch diameter, and are right-hand Langlay, special scale construction, of the best obtainable quality. The hoisting speed is 800 feet per minute.

The hoist is of a double-drum, post-brake pattern, driven by a 450-hp., 2,200-volt motor through a single reduction of herringbone gears. The brakes are controlled by oil under pressure, and are counterweighted to set automatically when the oil pressure is released. They are connected through a solenoid trip to Lilly controls, which automatically cut off the electric current and allow the brakes to set, in case of overspeeding or overwinding. The drums are 6 feet in diameter and are not grooved.

The skips discharge into a small steel transfer chute in the headframe, which is equipped with a heavy baffle to break the fall. The ore is loaded from the transfer chute into a shuttle car which travels on a trestle parallel to the long axis of the shaft and dumps into any one of the five storage bins. This car is of the gable-bottomed type, and has a capacity of 150 cubic feet. It is operated by a small electric hoist mounted at the and of the trestle. The five storage bins have chutes along the 0-meter haulage-way (fig. 8), equipped with check gates and underswung cut-off gates exactly like those used in the glory-hole loading stations on this level.

These bins provide a surface storage capacity of more than 60 per cent of the daily production. Tramming on the 0-meter level is thus nearly, if

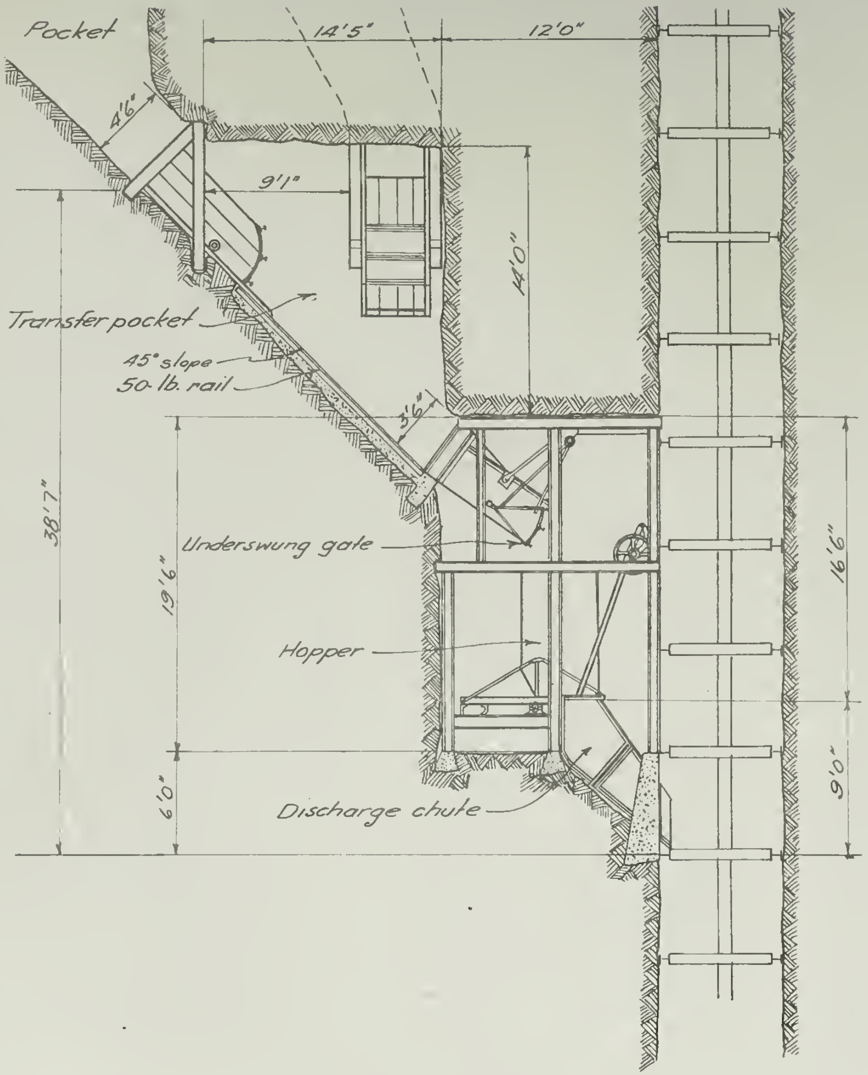


Figure 15.- Projection of loading pocket, 105-meter level

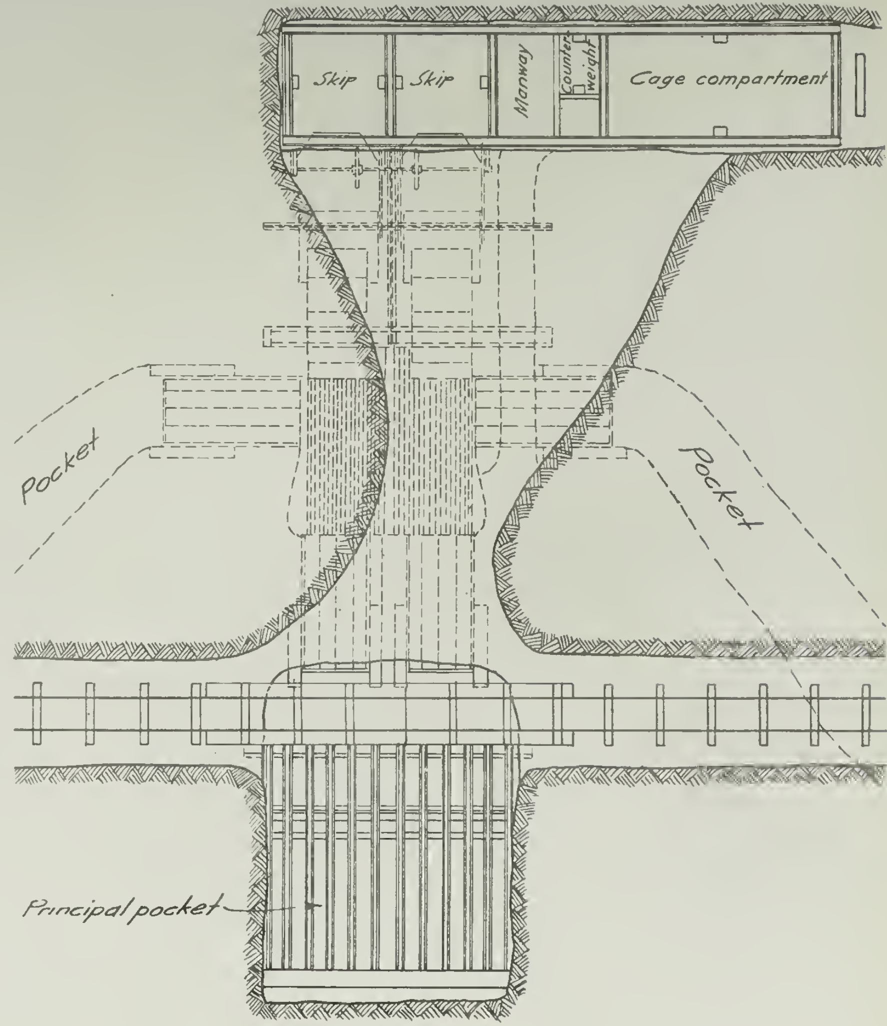


Figure 16.- Plan of loading pocket, 105-meter level

not quite, independent of the hoisting, and can be regulated to achieve the maximum efficiency of the trains and the crushing plant at the mill.

Ore from each orebody can be kept separate until delivered to the crusher. As the only reliable production sample that can be secured is the one cut by an automatic sampler at the mill after crushing, the separation of the ore until after this sample has been cut has been very valuable in furnishing a reliable check on the grade of ore from each of the several orebodies.

Figures 15 and 16 show the arrangement of pockets and skip-loading mechanism serving the 105-meter level. This installation was made in 1924, and still gives excellent service. The upper transfer pocket is essential where ore from different sources is to be kept separate. The hoppers are easily manipulated and rapid in action. The average halted time of the skip while receiving its load is 6 seconds. The chief criticism of the installation is that it operates with considerable violence, with a resulting high cost of maintenance. The retardation of the action by means of springs or counterweights would be desirable, but no practical system has been devised to secure this result. Access to the control gates of the upper pockets is had by way of a small crosscut from the shaft manway.

Figure 17 shows the corresponding installation to serve the 165-meter level. The hoppers were given a smaller horizontal cross section and a slope of 60° to render them less violent in action. The mechanism retains all the other features of the other installation. It is slightly slower, operates more smoothly, requires less maintenance, and is considered a better installation.

The hoisting capacity of the shaft has never been demonstrated over a long period but is estimated to be more than 1,700 tons per 8-hour shift from the 105-meter level pockets.

The service cage is 12 feet long and 5 feet wide. It weighs approximately 4 tons and is counterbalanced by a 5-1/2-ton counterweight. The hoisting speed is 600 feet per minute. The cage hoist is driven by a 150-hp., 2,200-volt motor through a single reduction of herringbone gears. The drums are 6 feet in diameter and are grooved. Sheaves and cables are the same as used on the skip hoist. Brakes are hand operated. The hoist is equipped with a Lilly control to prevent overspeeding and overwinding.

WAGE, CONTRACT, AND BONUS SYSTEMS EMPLOYED

The contract system has been generally in force on all parts of the work where measurable units of output for small groups of men can be accurately measured. Practically all development, preparation, and ore breaking are on this basis. Timbering of routine or standard nature is contracted wherever possible. Tramming, hoisting, and servicing of various kinds are done on day's pay.

country at the time; (2) insufficient supervision, due to the diversity of the operations and the wide areas covered by the foremen; and (3) the failure to enlist the entire cooperation of petty bosses and group leaders. Many of these latter became successful contractors when the contract system was restored.

VENTILATION

Natural ventilation of the workings has always been adequate, because of the existence of many old workings to which connections were made at frequent intervals.

FIRE HAZARDS

The principal fire hazard in the mine has been the top-slice area. Practically all of the workings in this area were timbered and the mat itself, of course, was an ever-present danger. It was recognized that a fire here would be disastrous. All of the workings in and around the area not actually required for ventilation or extraction of ore were tightly sealed off with adequate masonry bulkheads. Fire doors were constructed at stragetic points so that gases could be controlled in the event of fire. Fire extinguishers were kept at handy places in the area, and bosses and contractors were instructed in their use. Connections were made from the main air lines to the pump columns in the nearby pump shaft so that a large volume of water at high pressure would be at once available if required. The area was patroled by inspectors at all times when men were not working there; that is, on the graveyard shift and on Sundays and other holidays.

A large number of men were trained in the use of oxygen breathing apparatus, and four crews of five men each were kept in continued training. Demonstrated ability to use the apparatus has been a requisite for employment of a foreign boss in the mine. A smoke room was provided to make the training as effective as possible. Fire drills were held in the top-slice area at least once each month.

The top-slice operation was carried through to conclusion without a serious fire. The only threat of fire in the whole operation was a very small and smouldering blaze that resulted from blasting timber in a well-ventilated part of the slice. A wide-awake Italian boss extinguished the fire in a few minutes with the contents of three fire extinguishers. The area for 10 feet or so around the blaze was at once re-opened by spiling to insure that no sparks remained, and the area was kept under close observation for several weeks afterward, but nothing further occurred. The incident served as a warning of the ever-present danger, and gave a practical demonstration of the high value of careful and well-understood directions as to what to do in case of fire.

FIRST-AID ORGANIZATION AND ACCIDENT PREVENTION

Because of the extreme likelihood of the infection of open wounds in this country, first aid to injured men in the mine is reduced to the minimum of simple bandages, splints, and tourniquets, and a large number of workmen and all of the bosses have been trained to give this treatment. The policy is to get the injured man to the hospital with all possible dispatch. In serious cases, when it would be dangerous to move the injured man, the doctor or his assistant comes to the mine. Stretchers are kept at convenient places underground and first-aid material is kept in the principal magazines. A completely equipped first-aid station is maintained near the collar of the General shaft. The company maintains a complete and modern hospital.

Safety work is under the direction of a Mexican engineer. He has under him a corps of inspectors who are constantly in the mine, on the lookout for dangerous conditions. When found, such conditions are immediately reported to the boss directly in charge of the work. In urgent situations, the inspectors have authority to stop the work until the place has been inspected by the boss, who then assumes all further responsibility. The inspectors make daily reports to the safety engineer, who personally verifies special conditions and takes them up with the mine officials.

The use of goggles, hard-boiled hats, and shoes or sandals of special design is obligatory for all men underground. The use of canvas leggings is also required of men working on grizzlies or other places where leg protection is necessary. This material is supplied by the company at a price considerably below cost.

Each lost-time accident is personally investigated by the safety engineer and the foreman in charge of the work, usually accompanied by someone of higher authority. If possible, the accident is reenacted to determine the personal responsibility. Reports are made to the management, and are followed up in detail. Lay-offs are now applied to experienced men who are injured by their own carelessness, and to contractors, bosses, and others who countenance violation of safety regulations. This measure was adopted as being the only way of influencing those men who are not amenable to other methods; a thorough investigation is made of each case before laying off a man.

All bosses are paid a monthly safety bonus based upon the total shifts worked under their supervision and the number of accidents charged to their organizations.

Once each month the mine workings are thoroughly inspected by committees of workmen, accompanied by the safety engineer or his assistant. A written report is made of all conditions thought to be unsafe, and these reports are discussed at a monthly meeting of the committees, all of the foreman and bosses, and the mine officials. Each recommendation is given thorough consideration, and a definite decision is promptly reached. It is then the duty of the safety engineer to follow up on each accepted recommendation and see that is is carried out.

The contractors are called together once each month and the accident record for the preceding month is discussed with them in detail, particularly with reference to the accidents that occurred among their men and to measures that can be taken to avoid similar accidents in the future.

A general safety committee, comprising the management, department heads, and principal assistants, meets once each month for a review of the record of the previous month and for the discussion of safety matters of a more general nature.

By far the greater part of the accidents that occur are comparatively slight in their nature and can be traced directly to the following causes: (1) Inadequate clothing; (2) personal carelessness; (3) lack of intelligence or plain torpidity on the part of the injured man. Injured men receive full time while incapacitated, so that many of them do not give sufficiently serious regard to a minor accident involving a few days lost time.

At present, a special effort is being made to bring home to every man his responsibility for his own safety. This is being done by (1) determining the personal responsibility for the accident in every possible case; (2) by personal lectures to the men who are injured, by the safety inspector, the foreman involved, or in more serious cases, by the mine officials; and (3) by discipline in the form of lay-offs of from three to five days for injured men who are convicted of carelessness or indifference.

It is probable that future progress in accident prevention at Fresnillo must be along these lines.

Accident Statistics

Complete statistics to cover the early years of the operation are not available. For the year ended June 30, 1925, the accidents per 1,000 shifts worked in the Hilltop orebody were as follows:

Fatal	0.00
Grave	.03
Minor	.59
Total	0.62

The above figures cover all shifts charged to the Hilltop operation including transportation, surface, and general mine expense. Grave accidents were due to two principal causes: (1) Failure to keep the slopes of the glory holes carefully barred down; and (2) failure of workmen always to be tied with a rope when working in steep and dangerous places.

For the six years from July 1, 1925, to June 30, 1931, ore-breaking accidents per 1,000 shifts worked for the two principal mining methods were as follows:

	Glory holing	Top slicing
Fatal	0.0044	0.0048
Grave	.0286	.0095
Minor	.3722	.6762
Totals	0.4052	0.6905

The above figures apply only to ore breaking and do not include transportation, surface, or general mine expense. They exclude also accidents that do not occasion a loss of time for the injured man.

For divisions of the work other than ore breaking, the accident statistics for the oxide divisions can not accurately be separated from the total for the mine. The following tabulation of total accidents per 1,000 shifts worked for the whole mine, including the sulphide divisions, is therefore presented as showing the general trend.

Year ending June 30	Fatal	Grave	Minor	Total
1926	0.02	0.01	1.50	1.54
1927	0.01	0.01	1.42	1.44
1928	0.03	0.02	1.31	1.36
1929	0.03	0.06	1.26	1.35
1930	0.01	0.04	0.86	0.91
1931	0.03	0.03	0.62	0.68

The decided improvement in the records for the last two years is the result of a determined effort to reduce the number of accidents and to interest the workmen in their personal safety. The effects of a safety campaign are cumulative to a considerable degree. It has been difficult to establish a trend toward improvement, but once established, it is expected to continue.

Table 1. - Hilltop orebody

Glory-hole method. 3,858,688 tons of ore.

July 1, 1921 to June 30, 1925. 67,092 tons of waste stripping.

A - MINING COSTS (in U. S. dollars per short ton of ore)

	Labor	Super-vision	Power	Explo-sives	Timber	Other supplies	Total
Development ...	$0.0179	$0.0024	$0.0018	$0.0040	$0.0023	$0.0069	$0.0353
Ore breaking ..	.0574	.0032	.0190	.0510	.0009	.0182	.1497
Secondary blast	.0193	.0008	.0016	.0021	....	.0014	.0252
Transportation.	.0252	.0016	.0058	....	....	.0106	.0432
General under-ground expense	.0050	....	.0007	....	....	.0018	.0075
Surface expense	.0035	....	....	....	....	.0006	.0041
Totals	0.1283	0.0080	0.0289	0.0571	0.0032	0.0395	0.2650

Per cent of total cost

Labor and supervision	51.5
Power and supplies	49.5

B - MINING COSTS (in units of labor, power, and supplies, per short ton of ore or per foot of advance)

Ore production

Labor (man-hours per ton)	
Ore breaking	0.332
Secondary blasting	.129
Transportation	.129
General underground exp..	.023
Surface expense	.014
Total	0.627
Tons per 8-hour man-shift	12.76
Power (Kw.h. per ton)	
Ore breaking	1.41
Secondary blasting	.12
Transportation	.43
General underground exp..	.05
Total	2.01
Supplies	
Explosives, lb. per ton..	0.217
Timber, bd.ft. per ton ..	0.070

Development (313 tons mined per foot of advance)

Labor, man-hrs. per ft...	32.14
Power, Kw.h. per ft.	41.5
Supplies:	
Explosives, lb. per ft.	5.04
Timber, bd.ft. per ft..	15.9

The waste-to-ore ratio in the Hilltop orebody was 0.017 to 1. As the foregoing costs are calculated on the basis of tons of ore, the figures should be divided by 1.017 to obtain the cost per ton of material mined.

Table 2. - Orebodies other than the Hilltop

Orebody	Mining method	Period	Tonnages
Catillas and San Nicolas	Glory hole		4,544,914 1/
Pilar	Top slice		436,500
San Pedro	Miscellaneous	16 mos. 1926 and 1927	130,000

A - MINING COSTS (in U. S. dollars per short ton of ore)

	Labor	Supervision	Power	Explosives	Timber	Other supplies	Total
Development	$0.0125	$0.0107	$0.0075	$0.0126	$0.0171	$0.0383	$0.1887
Ore breaking:							
Glory hole ...	.1303	.0179	.0290	.0584	.0037	.0238	.2631
Top slice	.6179	.0179	.0707	.0574	.4100	.0258	1.1997
Miscellaneous.	.3673	.0179	.0707	.0654	.0045	.0242	.5500
Transportation..	.0874	.0071	.0181	......	......	.0269	.1395
Gen. underground expense	.0659	......	.0140	......	......	.0763	.1562
Surface expense.	.0089	......	.....	......	......	.0015	.0104

1/ Plus 1,328,360 tons of waste stripping.

Table 2. - Orebodies other than the Hilltop--Continued

B - MINING COSTS (in units of labor, power, and supplies, per short ton of ore or per foot of advance)

Ore production

Labor (man-hours per ton)	
Ore breaking:	
Glory hole	0.800
Top slice	3.830
Miscellaneous	1,664
Transportation	.398
General underground expense	.311
Surface expense	.009
Power (kw. h. per ton)	
Ore breaking:	
Glory hole	2.13
Top slice	5.22
Miscellaneous	5.22
Transportation	1.33
General underground expense	1.03
Supplies	
Explosives (lb. per ton):	
Glory hole	0.343
Top slice	.338
Miscellaneous	.385
Timber (bd. ft. per ton):	
Glory hole	.082
Top slice	9.151
Miscellaneous	.010

The waste-to-ore ratio in the Catillas and San Nicolas orebodies was 0.292 to 1. As the ore-breaking costs given above for these orebodies (under "Glory hole") are calculated on the basis of tons of ore mined, those figures may be divided by 1.292 if it is desired to obtain the costs per ton of material mined. The costs shown for "Transportation," being calculated on an ore basis, may likewise be reduced by dividing by the factor 1.13 to obtain costs per ton of total material handled.

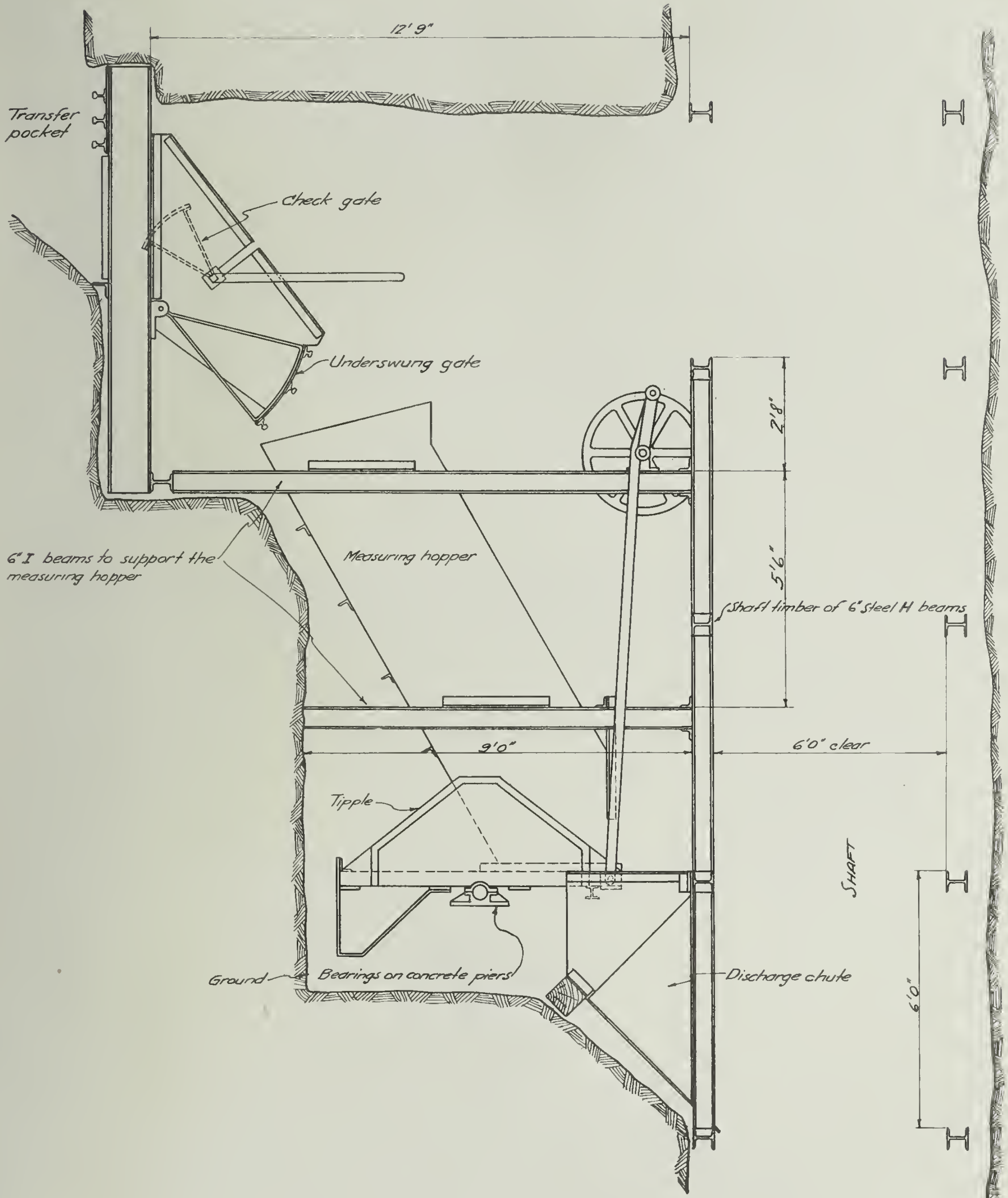


Figure 17.- Projection of loading pocket, 165-meter level

I. C. 6663 November, 1932

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

PRESSURE LOSSES DUE TO BENDS AND AREA CHANGES IN MINE AIRWAYS

BY

G. E. McELROY

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

PRESSURE LOSSES DUE TO BENDS AND AREA CHANGES IN MINE AIRWAYS[1]

By G. E. McElroy[2]

INTRODUCTION

The purpose of this paper is not to present new data on the pressure losses due to bends and changes of area in mine openings through which air flows, but rather to correlate existing data, determined mainly on small duct systems and expressed in a variety of ways, according to uniform methods that facilitate their use in pressure-loss computations. These are required in such air-flow problems as determining the quantity that will flow through mine openings under a certain pressure difference or, conversely, the pressure difference that will be required to maintain a certain rate of flow.

Air flowing through ducts, such as mine airways, encounters resistance to flow due to interference of the individual air streams with each other caused by the wall surfaces. The result is that part of the kinetic energy of the flowing stream is converted to heat; and constant renewal of the kinetic energy from the total energy of the flow is required to maintain it. This process manifests itself by a decrease of the total pressure (static pressure plus velocity pressure) of the air in the direction of flow; in order that flow may occur, there must necessarily be a difference in total pressure between the two ends of the airway equivalent to the pressure losses occasioned by the flow, which automatically adjusts itself in magnitude to the pressure difference maintained.

Pressure losses may be divided into two separate types, according to their mode of origin: "Friction" pressure losses caused by the drag of the walls on the air stream, which are dependent primarily on the condition of roughness of the individual wall surfaces; and "shock" pressure losses caused by changes in the area occupied by the air stream as affected by either deflection from a straight line or by changes of area of the airway, which are dependent primarily on the relative positions of the individual wall surfaces with respect to the lines of flow.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6663."

2 Mining engineer, U. S. Bureau of Mines.

SHOCK PRESSURE LOSSES

All departures from the condition of straight airway of uniform cross-sectional area result in pressure losses that are here termed "shock" pressure losses. This expression has heretofore had a more limited application, having been reserved for designating the pressure losses accompanying abrupt enlargement of the airway in the direction of flow and due to a faster stream expanding into a larger area and slower stream. The term has been selected for broader application because analyses of experimental results indicate that it designates the mode of action for all pressure losses other than friction losses; that is, they are due in all cases to shock loss resulting from a faster stream expanding into a slower stream. Although all shock losses may be considered as caused by changes in the area actually occupied by the flow, it is convenient to divide them into two general classes: Those caused by changes in direction of the airway, and those caused by changes of area of cross section of the airway.

General Formula for Shock Pressure Losses

Shock pressure losses have been found to bear a constant ratio to the velocity pressure corresponding to the mean velocity of flow - absolutely constant in the case of most conditions of area changes, and practically constant in the case of bends. They can be generally represented and computed, therefore, by means of the general formula,

$$H = X\,H_v \qquad (1)$$

where H is total shock pressure loss in inches of water,
H_v is the velocity pressure corresponding to the mean velocity of flow, in inches of water,
and X is an empirical factor based on experiments and termed the "shock factor."

Since H and H_v are always expressed in the same units and are equally affected by air density changes, X is independent of both density and the units used, in which respect it differs from the friction factor used in formulas for computing friction losses. It is the number of velocity pressures equivalent to the shock pressure loss. When it is desired to make a summation of individual shock losses that occur regularly along an airway, it is the number per unit of length, for which 100 feet will be used throughout this paper.

SUMMARY OF DATA AVAILABLE ON SHOCK FACTORS

The object of this paper is, therefore, to present the results of an examination and analysis of existing data on shock pressure losses at bends and area changes in terms of shock factors for application to mine airway conditions.

The examination reveals that, although we have no rational method of computing shock factors for bends, we do have a large number of direct determinations of factors for bends in ducts of uniform area for which the general relations are sufficiently consistent to permit their general application to bends in mine airways of uniform area. The magnitudes of the shock factors given by different sets of test data are so inconsistent, however, due to abnormal flow conditions in test installations, that their application to normal flow conditions in mine airways is subject to a large degree of approximation. No rational method of computing factors for bends in airways of nonuniform areas is available and but few directly determined factors. Unsymmetrical expansion is involved, for which we have no law, and the effect of nonuniform areas on the degree of contraction that takes place in the flow is also unknown. Bends between sections of different areas are a very common condition in mine airway systems, but we can not compute factors for them with any degree of certainty until more data are available.

For symmetrical area changes in mine airways, we not only have a rational method of computing shock factors for practically all of the conditions encountered, but also very good agreement in the data required for the computations. However, area changes in mine airways are rarely symmetrical and, as in the case of bends between different areas, we have no rational method of computing the shock factors for nonsymmetrical area changes and we have only a few directly determined shock factors for such conditions. Unsymmetrical expansion is also involved in this case, and the effect of lack of symmetry on contraction factors is also unknown. We can not compute shock factors for unsymmetrical area changes in mine airways with any degree of certainty, therefore, until more data are available.

Apparently, the standard formula for shock loss applies only to conditions of symmetrical expansion around the whole perimeter of the air stream and does not apply to conditions of unsymmetrical expansion, although it has always been assumed to hold for unsymmetrical conditions without, to the author's knowledge, any theory or test results to confirm the assumption. Until we have a verified formula for computing shock losses due to unsymmetrical expansion, we will have to be satisfied with direct determinations of shock factors limited in application to the exact conditions of the test installation. It is possible that a few comparatively simple tests would reveal the desired law, or that it may be deduced from available test results; also, it appears probable that it will be found to be connected to the law for shock loss for symmetrical expansion through some simple function of the proportional part of the perimeter involved in the unsymmetrical expansion and that contraction coefficients will also be found to be similarly related.

Shock Factors in Terms of Friction Factors and Equivalent Lengths

Friction pressure losses are of major importance in almost any air-flow system; the common disposition, therefore, has been to compute the shock losses with the friction losses by means of various conventions of a greater or less degree of approximateness. This procedure may be justified for duct systems

in which the shock losses are relatively unimportant and the wall conditions are very uniform, but a more exact procedure is desirable for mine airway systems in which the shock losses are relatively important and the wall conditions are rarely uniform.

Shock losses have been found to be dependent on the relative positions of the wall surfaces and independent of the degree of roughness of the surfaces. For this reason they can not be computed correctly as friction pressure losses, although methods for obtaining approximate accuracy over a limited range of conditions are available.

Many of the data available on shock losses are expressed in terms designed to facilitate their computation as friction losses - that is, as increments to the friction factor or as equivalent lengths of airway expressed in feet or diameters.

It is therefore desirable to develop the formulas required for obtaining shock factors for shock pressure losses expressed in these forms.

A standard formula for computing friction pressure losses is

$$H_f = \frac{K\ P\ L\ V^2\ d}{5.2\ A\ \ 0.075} \qquad (2)$$

where H_f is total friction pressure loss, in inches of water,
K is an empirical factor for air having a density of 0.075 pound per cubic foot and termed the "friction factor,"
P is perimeter of cross section in feet,
L is length in feet,
V is mean velocity of flow in feet per minute,
d is weight of air in pounds per cubic foot,
and A is area of cross section in square feet.

A standard formula for computing velocity pressure is

$$H_v = 0.0^6831\ V^2\ d \qquad (3)$$

where H_v, V, and d have the same designations as before.

Increments to Friction Factor.- Shock losses that are regularly intermittent are computed along with the friction losses by increasing the friction factor used enough to make the result of the computation cover the total of both types of pressure losses.

If K' denotes the friction factor estimated so as to include a shock pressure loss of X velocity pressures imposed upon each 100 feet of an airway for which the strictly friction pressure loss is that corresponding to a friction factor K, then K' = K + k, where k is the increment to the normal friction factor required for including the shock pressure loss. By transposing (2) and expressing velocity in terms of velocity pressure from (3), we have, for the increment to K at standard air density conditions of 0.075 pound per cubic foot,

$$k = \frac{X\,H_v\,5.2\,A\,0.075}{L\,P\,V^2\,d} = \frac{X\,H_v\,5.2\,A\,0.075}{100\,P\,d\,\dfrac{H_v}{0.0^6831\,d}} = 0.0^8324\,\frac{A}{P}\,X \qquad (4)$$

The increments to K thus vary directly as the $\frac{\text{area}}{\text{perimeter}}$ ratio and, since this ratio is largely a matter of the actual size of the airway, the variation in increments for constant values of X per 100 feet is large. For a 6-inch circle or square the A/P ratio is 0.125, for a 6-foot circle or square it is 1.5, and for a 20-foot circle or square it is 5.0. However, the A/P ratios for mine airways of the sizes normally encountered range only from about 1.0 to 2.0, with an average of approximately 1.5, for which the increment to the friction factor is about 0.0^85 times X per 100 feet of airway. Compared to the normal friction factors, the increments are relatively small for rough-walled mine airways and the values of X available offer but an approximate degree of selection, so that the possible accuracy of results of calculations is little affected by this method of computing shock losses if it is limited to the ordinary range of sizes of mine airways. For more accurate results, and for airways or ducts that do not fall within this area range, the shock losses should be separately computed, or, if computed by use of the friction loss formula, the increments to the normal values of K should correspond to the actual A/P ratios.

Equivalent Length in Feet.- In computing the total pressure loss by means of the friction formula shock losses that are strictly local are allowed for by arbitrarily increasing the length over the actual length a sufficient amount to make the result agree with the total of both types of pressure loss. The increment in length required is often given in terms of the equivalent number of diameters rather than directly in feet.

The expression of shock losses in terms of equivalent feet of airway requires that the actual length of airway be increased by a length, L, that would have a friction loss equivalent to the shock loss XH_v, the length used for computation in the friction loss formula being the actual length plus L. By transposing the friction formula and expressing velocity in terms of velocity pressure, we have, for the increment L at standard air density,

$$L = \frac{X\,H_v\,5.2\,A\,0.075}{K\,P\,V^2\,d} = \frac{X\,H_v\,5.2\,A\,0.075}{K\,P\,d\,\dfrac{H_v}{0.0^6831\,d}} = 0.0^6324\,\frac{A}{K\,P}\,X \qquad (5)$$

Since the equivalent length in feet varies both directly as the $\frac{\text{area}}{\text{perimeter}}$ ratio and inversely as the friction factor, the range of increments for constant values of X is so large that approximate accuracy of direct application of test data is possible only for a very limited range of values of K and A/P. For the limited range of average mine airway conditions, the equivalent length in feet varies from about 300 X to 15 X.

Equivalent Length in Diameters.- The expression of shock losses in terms of equivalent diameters of airway or duct merely means that the increment of length, L, is divided by the diameter, D. The ratio A/P can also be expressed in terms of the diameter, D, thus:

$$\frac{A}{P} = \frac{\frac{\pi D^2}{4}}{\pi D} = \frac{D}{4} \qquad (6)$$

Dividing (5) through by D and substituting D/4 for A/P, we have,

$$\text{Equivalent diam.} = \frac{L}{D} = \frac{\frac{0.0^6 324}{K}\frac{A}{P}X}{D} = \frac{\frac{0.0^6 324}{K}\frac{D}{4}X}{D} = \frac{0.0^7 81}{K}X \qquad (7)$$

The equivalent number of diameters is thus independent of the A/P ratio and, for constant shock factors, varies inversely as the friction factor. The range of direct application of test data is therefore limited only to corresponding degrees of roughness of airway, or duct. Such an expression of shock losses is therefore useful in those fields of ventilation where ducts of approximately uniform conditions of roughness are the only airways involved, but it is not applicable to mine ventilation conditions of very variable conditions of roughness, as expressed by friction factors ranging from $0.0^8 20$ to $0.0^7 20$ for conditions commonly encountered, and for which the equivalent diameters may vary from about 40 X to 4 X.

Conversion Chart.- A graphic chart for converting shock factors to equivalent increments to friction factor or length, or vice versa, is presented in Figure 1. Slide-rule computations, based on formulas (4), (5), and (7) are the most simple and rapid means of converting factors, but the chart offers a rapid independent check of results.

The chart appears complex because three charts have been combined into one, but it is easily used, offering simply a graphic method of performing the multiplications and divisions designated by the three formulas, with the scales of values arranged to correspond. The solutions of three examples - the conversion of X to K, L and $\frac{L}{D}$ for a particular set of conditions - are shown on the chart by dotted lines, with arrows to show the methods of procedure. These are also shown by the three thumb-nail sketches in the lower right-hand corner of the chart. Although separate scales of A (area) and P (perimeter) are provided for the graphic determination of the $\frac{A}{P}$ ratio, the latter appears as a separate scale which can be used directly with approximate accuracy by means of the parallel scale of equivalent D's, or size of equivalent circles or squares.

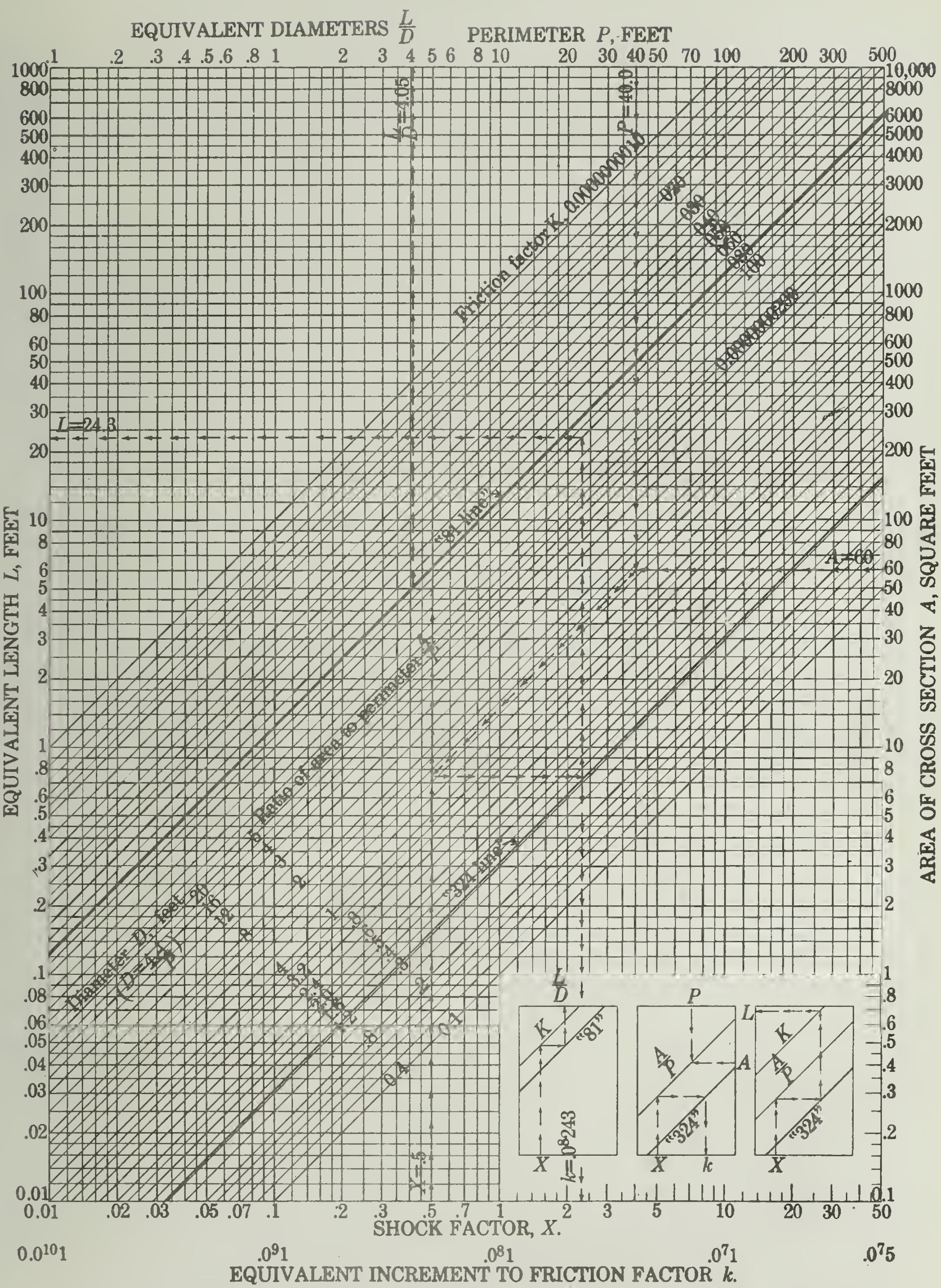

Figure 1.- Chart for determining corresponding values of shock pressure losses expressed in terms of shock factors, increments to friction factors, or equivalent lengths in feet or diameters

In the examples solved $A = 60$ square feet, $P = 40$ feet, $K = 0.0^{7}1$ and $X = 0.5$. Corresponding values of k, L and L/D are determined from the chart to be $0.0^{8}243$, 24.3 feet, and 4.05 diameters respectively.

Shock Factors for Bends

The effects of deflections from a straight line are important in their effect on mine airway pressure losses, since absolutely straight airways, except in the particular case of shafts, are the exception rather than the rule. The normal average variation of the exact center of successive sections from the straight line averaging such points - that is, the average irregularity (alignment in the case of timber-lined airways) of so-called "straight" sections - should properly be included in this class of losses, but the effects are small and not separable from the friction pressure losses with which they are always included in determining the range between minimum and maximum values of friction factors for clean, straight airways.

It is important to note that the angle of a bend is the angle through which the air is deflected - that is, 180° minus the angle as commonly designated; thus airways joined at a 135° angle produce a 45° bend.

Many conditions of the air flow, such as velocity, area, density and viscosity, which have been found to affect the constancy of friction factors to a greater or less degree dependent on the range of conditions considered, have somewhat similar effects on the constancy of shock factors for bend losses, but the effects are so small, in comparison with the uncertainty of application of existing data to the irregular conditions presented by mine airways, that they will be ignored here.

Flow Conditions at Bends

Experimental determinations of shock losses at bends indicate that the air crowds to the far side of the bend and occupies less than the full area available at the departure end of the bend; and that the excess of pressure loss, over that due to friction as determined by the rubbing surface, is a shock loss due to abrupt one-sided expansion from this contracted area to the full area following the bend, as pictured in Figure 2. The degree of contraction in the area occupied by the flow at the departure end of the bend is apparently determined not only by the position of the walls of the airway throughout the bend, but also by the distribution of velocities over the cross section at entrance to the bend. The expansion is made at a small angle and thus takes place over a considerable length of airway downstream of the bend, depending on the degree of contraction preceding the expansion; and throughout this section not only are static pressure measurements unreliable, but the average velocity pressure can not be determined from area and quantity data, since the air stream does not occupy all of the cross section of the airway. Where expansion takes place from the contracted stream direct to the atmosphere, as from an elbow on the end of a duct, the shock loss is much larger than that for abrupt expansion to the airway area only on account of the greater difference in the velocities involved. The ratio of the shock losses for these two conditions of installation

of the same bends is reported to be practically constant regardless of the type of bend; various sets of tests give figures (not corrected for included friction losses) ranging from 56 to 67 per cent for the loss with following duct in terms of the loss without following duct. This difference of shock pressure losses for the two different conditions of installation is often explained as due to "conversion" of pressure following the bend, whereas it results only because of the difference in conditions of shock loss.

The abrupt expansion in the area occupied by confined flow following a bend is not symmetrical but one-sided, or at least affects but part of the perimeter of the section, and the standard formula for shock loss - as equal to the velocity pressure corresponding to the difference of the velocities involved - apparently does not apply. However, it is a simple matter to determine the shock loss directly by test and express it in terms of velocity pressures, or shock factors.

In mine airways, the area following the bend is rarely the same as the area preceding the bend, a condition that affects the degree of contraction and the shock loss. In order to calculate the shock losses for such conditions it is needful to know not only the effect of nonuniform area conditions on the contraction coefficient, or ratio of contracted area to area at entrance, but also a formula for the shock loss due to unsymmetrical expansion. Until we have these data, the calculation of pressure losses for bends in airways of nonuniform area must be approximate at best. With few exceptions, therefore, our data on shock factors for bends are limited to conditions of uniform area of cross section.

Sinuosity, Crookedness, and Curvature

Mine airways often include numerous small bends and curves, involving deflections of 30° or less, and are termed "sinuous," "crooked," or "curved," as the case may be. Drifts following a vein and alternately turning one way and then the other may be "sinuous" if largely composed of short curved sections or "crooked" if of short straight sections. Where the deflections are approximately constant and in the same direction, there are "continuous curves" or "turns." The conditions are usually very irregular and the individual shock pressure losses are so small as to defy calculation, but a very rough approximation may be made of the combined shock losses from existing data[3], as follows:

Slight degree; $X = 0.2$ per 100 feet. A large-radius curve, a sharp bend of about 15° deflection, or a wall-line close to the center line, where they occur separately and not oftener than once every 100 feet; or, a small-radius curve, a sharp bend of about 30° deflection, or a wall-line crossing the center line, where they occur separately and not oftener than once every 200 feet.

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3 McElroy, G. E., and Richardson, A. S., Resistance of Metal-Mine Airways: Bull. 261, Bureau of Mines, 1927, p. 131.

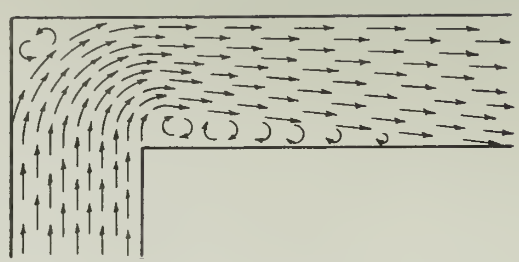

Figure 2.- Sketch showing general conditions of flow at bends in airways

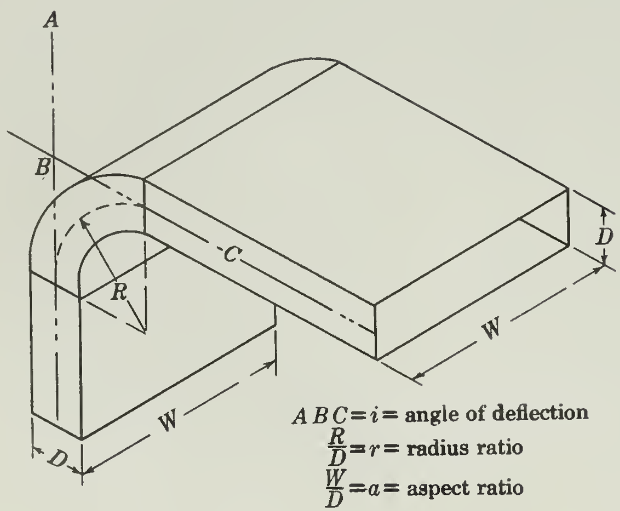

Figure 3.- Sketch defining bend characteristics

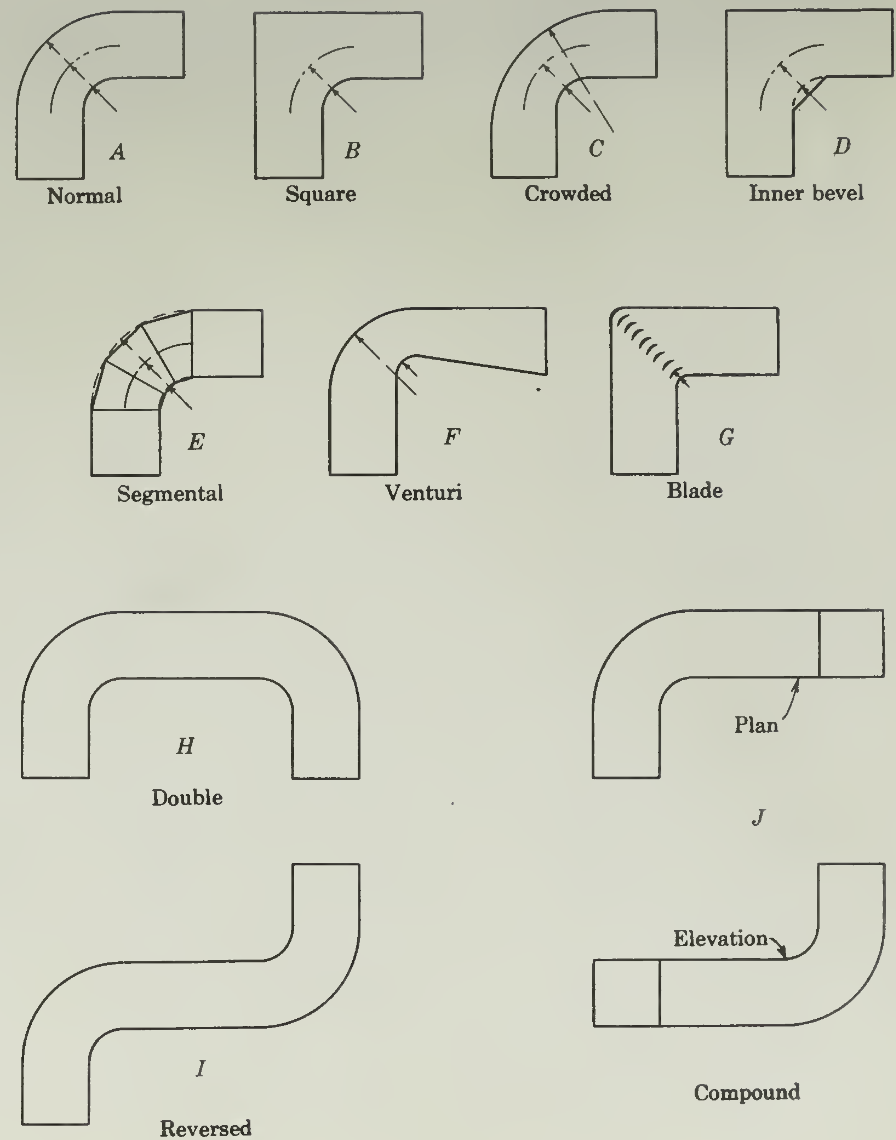

Figure 4.- Common types of right-angle bends

Moderate degree; X = 0.3 per 100 feet. Conditions intermediate between slight and high degree.

High degree; X = 0.5 per 100 feet. A continuous large-radius curve, a continuous turn of repeated small deflections of 10° to 15° every 20 to 30 feet, bends of 20° to 30° deflection every 50 to 100 feet, or a wall-line crossing the center line about every 50 feet.

The foregoing descriptions are rather meager for lack of comprehensive test data, but they are sufficient to permit of approximate computations for mine airways, which is about all that the variable area conditions of such airways usually justify.

Single Right-Angle Bends

Deflections of 90°, or right-angle bends, in ducts are commonly designated as elbows, and most of the data available on bend losses are the result of laboratory tests[4] on elbows in small ducts made under conditions that facilitated the test work without necessarily producing normal flow conditions at the entrance to the elbow or complete expansion of the flow beyond the elbow; and many of the published results include friction losses for four to ten or more diameters of test duct - elbow and following duct of varying length - involved in the test installation. A few results for right-angle bends in mine airways[5] are available, but these results are somewhat uncertain on account of the variable areas of airways involved in test zones and the details of the required construction.

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4 Harding, Louis A., The Flow of Air in Heating and Ventilating Ducts: Trans. Am. Soc. Heat. and Vent. Eng., vol. 19, 1913, pp. 219-227.
Bussey, Frank L., Loss of Pressure Due to Elbows in the Transmission of Air Through Pipes or Ducts: Trans. Am. Soc. Heat. and Vent. Eng., vol. 19, 1913, pp. 367-376.
Willard, A. C., Report on Hudson River Vehicular Tunnel Ventilation Investigation, by the University of Illinois Engineering Experiment Station, for the New York State and New Jersey Interstate Bridge and Tunnel Commissions: prepared Jan., 1922, but not published.
McElroy, G. E., and Richardson, A. S., Experiments on Fan-Pipe Installations at Butte, Montana: Rept. of Investigations 2509, Bureau of Mines, July, 1923, pp. 10-11.
Wirt, Loring, New Data for the Design of Elbows in Duct Systems: Gen. Elec. Rev., vol. 30, 1927, pp. 286-296.
Cooke, W. E., and Statham, I. C. F., The Resistance to Flow of Air at Bends and in Straight Airways: Trans. Inst. Min. Eng. (London), vol. 77, 1929, pp. 188-212.

5 Greenwald, H. P., and McElroy, G. E., Coal-Mine Ventilation Factors: Bull. 285, Bureau of Mines, 1929, pp. 66-69.
McElroy, G. E., and Richardson, A. S., Resistance of Metal-Mine Airways: Bull. 261, Bureau of Mines, 1927, pp. 133-134.
Cooke, W. E., and Statham, I. C. F. Work cited.

The common type of bend in mine airways is one having sharp intersections of the walls at the inner and outer corners, a condition that gives a maximum shock pressure loss. Bends in ducts usually have the inner and outer corners rounded on concentric curves because it has been found that the shock loss decreases as the degree of curvature is increased. The degree of curvature is commonly designated by the ratio of the center line radius to the width in the plane of turning, termed the "radius ratio," or R/D in Figure 3. Just as it is harder to bend a plank on the wide dimension than on the narrow dimension, so it is easier as respects shock loss to have the air flow turn through the larger dimension of the airway than through the smaller dimension. In rectangular airways, therefore, the ratio of the side on which turning takes place to the other dimension is termed the "aspect ratio," or W/D in Figure 3, a term borrowed from aeronautics by Wirt, whose tests are about all the data we have on this factor of bend losses.

A careful consideration of all the available data on the effect of the radius ratio shows an extremely large variation in the results reported by different investigators, which is approximately 100 per cent even after corrections have been made for included friction pressure losses ($X = 0.02$ per diameter approximately for smooth ducts) and aspect ratio variations. Analyses of test installation conditions reveal causes for many of these seeming inconsistencies but no way of correcting the data to a basis comparable to normal air flow conditions both preceding and following the bend. Wirt's careful provision of uniform-velocity distribution at the elbow entrance, although provided to insure accuracy, apparently had the opposite effect and destroyed not only the absolute accuracy of his results for application to normal flow conditions of nonuniform velocity distribution, but even their relative value in at least one instance. Wirt found that a square outer corner had about 10 per cent less shock loss than a rounded outer corner and this fact was widely acclaimed in the technical press as a startling example of a case where test results refuted ordinary reasoning. The result is startling but apparently due to the test condition of uniform velocities at entrance to the elbows since many other tests, for normal flow conditions, show that the shock loss for an elbow with square inner and outer corners is about 20 per cent greater than for one having a square inner corner and rounded outer corner of conventional type; and the bureau's tests in coal-mine airways show that the difference in shock loss for the two conditions increases as the radius of the inner corner is increased. However, Wirt's tests developed considerable data on flow conditions in elbows and, of greater practical importance, the relative effect of the aspect ratio, which he put to immediate practical application in the development of the "blade" turn, a form of elbow in which curved blades inserted in a square-cornered elbow divide the single turn into the equivalent of a series of turns with radius ratios and aspect ratios favorable to low shock losses.

Effect of Radius Ratio

For the normal type of 90° bend, Figure 4 A, where the inner radius is one-half the width less than the center line radius and the outer radius is one-half the width greater, the shock loss due to the bend apparently varies inversely as the square of the radius ratio and, for an aspect ratio of one (round

or square section), the shock factor for this normal type of right-angle bend may be taken as approximately:

$$X = \frac{0.25}{r^2} \qquad (8)$$

where r is the radius ratio, or R/D in Figure 3.

The constant 0.25 is taken as a fair approximation of the most reliable test results for normal flow conditions in round and square airways. Separate sets of data indicate constants varying from 0.18 to 0.40.

The minimum value of r is 0.5 for the condition of square inner corner and an outer radius equal to the width, and X = 1.0. Good results are obtained where r = 1.5 and there is little practical advantage in exceeding r = 2.0 -- that is, a center-line radius greater than twice the width or diameter. For the "square" bend, Figure 4-B, r is also 0.5 where both inner and outer corners are square and most of the test results on small ducts indicate an increased shock loss over the normal type of rounded outer corner, and a shock factor of about X = 1.20. The bureau's tests on mine airways also show that, as the radius of the inner corner increases with the outer corner square, the shock loss decreases only about as fast as the value of r increases, or that

$$X = \frac{0.60}{r} \qquad (9)$$

for this special case of square outer corner (figure 4-B).

For the case of the "crowded" elbow (fig. 4-C) where the outer radius is made shorter than the normal concentric design - that is, less than one width longer than the inner radius - Willard's tests show that the pressure loss, while still varying inversely as the square of the radius ratio, is larger than for the normal design; 40 per cent larger when the outer radius is but 75 per cent of the normal radius concentric with the inner radius.

The bureau's tests with the inner corner of the airway cut off on a 45° bevel and outer corner square (fig. 4-D) a common condition at the top of mine air-shafts - show that the loss is about the same as for a square bend with a radial curve on the inner corner touching the walls at the same points as the bevel. A few tests on segmental elbows (fig. 4-E) indicate that the losses can be computed on the basis of the circumscribed radii, with a small allowance for "crowding" on the outer corner.

When it is desired to reduce the loss at a bend where a square inner corner can not be displaced, a form of construction based on the fact that the air crowds to the outer two-thirds of the bend in turning, and termed a "Venturi" bend, may be used. As shown in Figure 4-F, the area at the bend is smoothly reduced to about two-thirds, and is gradually brought back to normal area following the bend. According to Bussey, this form was suggested by Konrad Meier in his book "Mechanics of Heating and Ventilating." It has been the subject of a number of tests which indicate that the losses are approximately equivalent to those for a normal bend with a radius ratio of about 0.8 to 0.9, depending on the details of construction.

Effect of Aspect Ratio

Wirt's test data give the effect of aspect ratio on shock pressure losses for single right-angle elbows in terms of a factor expressing the relation of the loss at other aspect ratios to that with aspect ratio of 1. For ratios greater than 1, his data indicate that the shock factors vary approximately inversely as the square root of the aspect ratio. For ratios less than 1, his test data show a rapid departure from the curve expressing the foregoing relationship, but considering the latter as a possible result of uniform-velocity test conditions it seems best to hold to a simple relation between aspect ratio and shock factor until more data are available, and the effect of aspect ratio may be found approximately from the formula,

$$X = \frac{X'}{\sqrt{a}} \qquad (10)$$

where X' is the shock factor for the same type of bend with an aspect ratio of 1 (square section)
and a is the aspect ratio, or W/D in Figure 3.

Bends with aspect ratios greater than 1 have been termed "bucket" bends, and those with a ratio less than 1, "flat" bends. Bucket bends are always to be preferred, but it is a fact that the bend at the top of most mine air-shafts will be found to be a flat bend rather than a bucket bend, even though the increase in power cost in most cases greatly exceeds the decrease in construction cost, except for temporary installations.

Effect of Vanes

A number of tests have been made in which vanes have been installed at right-angle bends for reducing the shock loss. Such vanes introduce some resistance to flow by their added friction surfaces and obstructing edges, but act to divide the single bend into a number of separate bends, each deflecting a proportional part of the total flow, that have better radius and aspect ratios than the original single bend; and the net result is decreased shock loss. The vanes are commonly installed concentrically to the radius of the inner corner, and tests have shown that the best construction for a single vane in the normal type of bend is about one-third of the width from this corner. The probable effect of a vane, or vanes, can be roughly determined by calculation from radius and aspect ratios, but as allowance must be made for the increased losses due to the added rubbing surfaces and obstructions, the test results check calculated losses relatively rather than absolutely.

A good example of the use of vanes in improving radius and aspect ratios and thereby reducing the normal loss is the "blade turn" (fig. 4-G) developed by Wirt, in which short curved blades are installed centered on the diagonal of the turn and divide it into the equivalent of numerous bends with high radius and aspect ratios. With an elbow that had a radius ratio of 0.67 and an aspect ratio of 1.0, the installation of 11 short, curved, sheet-metal vanes, which divided the turn into 12 equal segments having radius ratios of 1.5 and

aspect ratios of 12, reduced the (abnormally high) shock loss of 0.90 velocity pressure (X = 0.90) to 0.22 velocity pressure (X = 0.22).

The addition of straight extensions, either upstream or downstream, to radial vanes has always shown an increased loss, in experiments, over the condition of radial vanes without such extensions.

Closely-Spaced Right-Angle Bends

Since the expansion of the contracted air stream at the departure side of a bend takes place over a considerable distance downstream from the bend, it is only natural that tests on closely spaced bends - bends so close to each other that the opportunity for full expansion is lacking - should show test results differing from those of single bends, as the case actually is. The same effect has been noted in aeronautics and other forms of air flow and is often referred to as the "shielding effect" of one object on another.

From the bureau's test data on closely-spaced right-angle bends in the same plane (Bull. 285, pp. 80-89), it appears that where the deflection of both bends is in the same direction the lack of complete expansion at the first bend not only reduces its shock loss below normal, but, by providing higher than normal velocities at the outer corner of the second bend, also results in a lower than normal degree of contraction at the departure from the second bend and thus reduces its shock loss to less than the normal. This combination has been termed a "double" right-angle bend (fig. 4-H).

Where the flow is deflected at the second bend in a direction opposite to that of the first bend, the combination is termed a "reverse" bend (fig. 4-I). Both Bussey's and the bureau's test data indicate that while lack of complete expansion may reduce the loss at the first bend, that of the second bend is increased by the higher than normal velocities at its inner corner, with the result that the total shock loss for a reverse bend is greater than the sum of the normals for both bends.

The downstream distance required for full expansion is not known, but the maximum is probably at least 10 diameters or widths and may be more, and is affected to some extent by the velocity of flow. Bureau tests in coal-mine airways 6 feet high and 9 feet wide with right-angle bends on 50-foot centers showed about 20 per cent greater loss for square-cornered reverse bends than for double bends of the same type, indicating that the expansion was still far from complete in 4.6 widths for that high rate of pressure loss. The same series of tests showed, by rough segregation of the losses for both bends of a double right angle, that the distance required for full expansion is materially decreased as the rate of pressure loss decreases, as would be expected, since the rate of pressure loss is dependent on the degree of contraction.

Where two closely spaced bends are not in the same plane, the combination is referred to as a "compound" bend (fig. 4-J). Here the higher than normal velocities at entrance to the second bend, resulting from incomplete expansion following the first bend, affect both inner and outer corners of the second bend equally. Although no complete test data are available, it can be assumed that the loss at the first bend would be lower than the normal and that of the second bend higher than the normal; and that, in all probability, the total loss for the combination would be approximately equal to the sum of the normal losses for the two single bends.

Effect of Angle of Deflection

Few test data are available for deflections other than 90°. Weisbach's[6] rather meticulous formula for square-cornered bends,

$$X = 0.9457 \sin^2 \tfrac{1}{2} i + 2.047 \sin^4 \tfrac{1}{2} i, \qquad (11)$$

indicates that the shock pressure loss varies practically as the 2.38 power of the angle of deflection, while the recent data of Cooke and Statham[7] for square-cornered bends indicate that it varies almost exactly as the square of the angle of deflection; and scattered test data also indicate that the square relation holds. Cooke and Statham's data on bends having radius ratios of 1 to 4, and thus low rates of pressure loss, give variable relations, but an analysis and comparison of their data indicate the strong probability that this result is due to inadequate deductions for friction pressure losses. It therefore seems best to assume that the effect of the angle of deflection on the shock loss may be determined approximately by the formula,

$$X = \frac{i^2}{(90)^2} X' = \frac{i^2}{8100} X', \qquad (12)$$

where X' is the shock factor for a right-angle bend of the same design, radius ratio, and aspect ratio,

and i is the angle of deflection, or exterior angle of the center lines, in degrees.

General Formulas for Shock Factors for Uniform-Area Bends

Since we have no reliable data for the effects of radius ratio and aspect ratio for deflections of other than 90°, we can only assume that the relative effects would be the same as for 90° deflections. From the incomplete and inconsistent data now available, it would appear that best complete formulas for computing approximate shock factors for shock losses due to bends in uniform-area mine airways are,

6 Peele, Robert, Mining Engineer's Handbook: 1st ed., New York, 1918, p. 1031.

7 See reference 2.

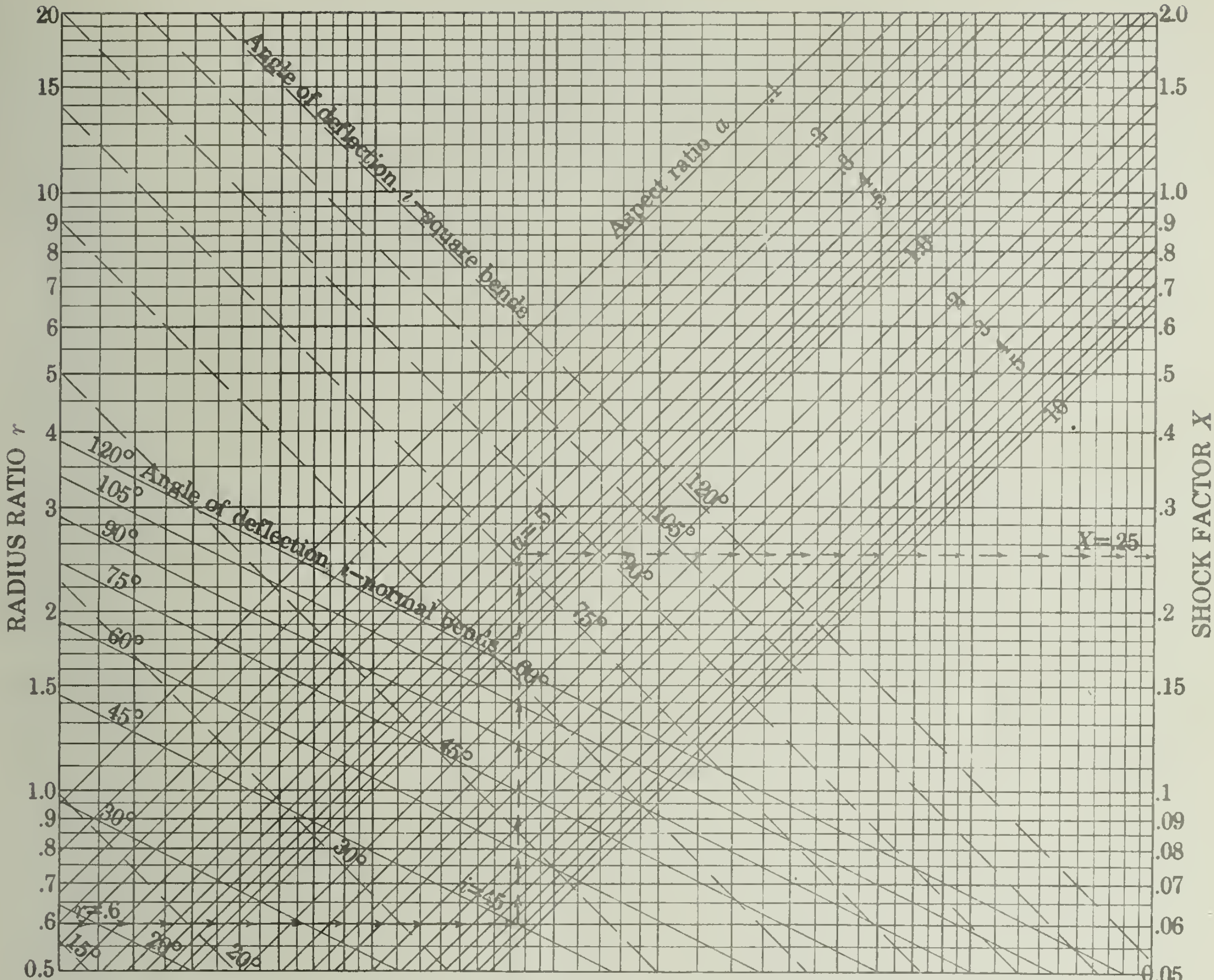
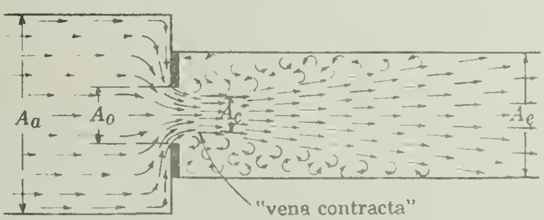


Figure 5.- Chart for determining shock factors, or shock pressure losses in terms of equivalent velocity pressures, for bends in airways of uniform area



$\frac{A_c}{A_o} = c =$ coefficient of contraction

$\frac{A_o}{A_a} = N_c =$ ratio of contraction (areas)

$\frac{A_o}{A_e} = N_e =$ ratio of expansion (areas)

Figure 6.- Sketch showing general conditions of flow and defining characteristics of area changes in airways

$$X = \frac{0.25\ i^2}{r^2 \sqrt{a}\ 8100} \qquad (13)$$

for the normal type of concentric-radii bend, and

$$X = \frac{0.60\ i^2}{r \sqrt{a}\ 8100} \qquad (14)$$

for "square" bends, or bends with a square outer corner. These reduce to

$$X = \frac{0.25}{r^2 \sqrt{a}} \qquad (15)$$

and

$$X = \frac{0.60}{r \sqrt{a}} \qquad (16)$$

for right-angle bends in rectangular airways, and to

$$X = \frac{0.25}{r^2} \qquad (17)$$

and

$$X = \frac{0.60}{r} \qquad (18)$$

for right-angle bends in square or round airways. Little is known regarding the factors for abnormal types, although a few such types have been discussed in this paper under "Effect of Radius Ratio." For the most common type of abnormal bend encountered in mine airways - that where the areas at entrance and departure are unequal - we unfortunately have no test data on which to base a rational method of computing the shock factors for the shock losses.

Chart for Determining Shock Factors for Bends

In Figure 5, a graphic chart is presented for the rapid determination of shock factors for shock losses due to "normal" and "square" bends in mine airways of uniform area. This is based on formulas (13) and (14). The method of using the chart is shown by the dotted-line solution of an example, with arrows indicating the direction of procedure. In the example solved in the chart, it is shown that, when the radius ratio of a normal bend is 0.6, the angle of deflection 45°, and the aspect ratio 0.5, the shock factor is 0.25. In using the chart, it is important to note that the full lines for angle of deflection should be used for the normal type of concentric-radii bend, whereas the dash lines should be used for bends with square outer corner.

Shock Factors for Area Changes

The area of the cross section of airways or ducts changes frequently in any ventilating system and each change requires the interchange of pressure from the velocity pressure form to the static pressure form, or vice versa. These interchanges are not 100 per cent efficient and some of the total pressure of the air stream is lost at each change, the magnitude of the loss depending primarily on the degree of abruptness of the change in area of the air stream, which may take place because of changes in the dimensions of the airway itself, or because of conditions that cause the air flow temporarily to occupy only part of the total area of the airway.

The shock pressure losses caused by area changes have been found to bear a constant ratio to the velocity pressure; absolutely so for the more abrupt changes such as orifices and practically so for the less abrupt changes. For the latter, slight variations, similar to those found with friction factors and shock factors for deflections, accompany variations in flow conditions, but they are not of sufficient magnitude to be taken into consideration here.

The pressure losses due to the normal average change in area between successive sections of an airway should properly be included, or allowed for, on a shock pressure loss basis, but the effects are small and not separable from the friction losses with which they have always been included in determining the range between the maximum and minimum values of the friction factors for clean straight airways; and there is no doubt that area changes have much to do in determining the spread between maximum and minimum values assigned to friction factors.

Intermittent Obstructions

Mine airways often contain intermittent obstructions that change the area of cross section under conditions where the individual shock losses are so small that they practically defy computation and where it is more convenient to determine directly the average total shock loss per unit of length, which the data[8] available permit with a very rough degree of approximation, as follows:

Slight degree: $X = 0.1$ per 100 feet; trolley box, water box, large flanged pipe, occasional small falls of roof, occasional small crossbars, hangers, props, etc.
Moderate degree: $X = 0.3$ per 100 feet; large fan pipes or tubings, occasional large crossbars or heavy hangers, frequent small falls, occasional constrictions, etc.
High degree: $X = 1.0$ per 100 feet; combinations of obstructions given above, large falls, occasional storage piles of timbers or pipes, closely set crossbars, props or constrictions, etc.

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8 McElroy, G. E., and Richardson, A. S., Resistance of Metal Mine Airways: Bull. 261, Bureau of Mines, 1927, p. 134.

These values are for average areas determined by subtracting the area occupied by the obstruction where the latter is continuous and of appreciable size. For the very high degrees of obstruction occasionally found in mine airways, it is impossible to set even approximate limits, as the normal characteristics become of minor importance in comparison with the characteristics of the obstructions and each case must be computed by analyzing its separate features and separately calculating the friction and shock losses involved.

Formulas for Shock Factors for Symmetrical Changes of Area

The available test data indicate that the pressure losses for area changes are practically all due to shock pressure loss caused by a faster stream expanding into a larger area and slower stream; there appears to be no perceptible pressure loss in converging air streams due to the converging itself, but the converging of the air stream causes it to contract beyond the edge of the constriction, giving rise to a condition at constrictions in area known as the "vena contracta," as illustrated in Figure 6. The area of cross section of the air stream is a minimum immediately following the edge of the constriction, and the shock loss is that caused by expansion from this area to the full area of the following duct with consequent lowering of the velocity and imperfect conversion of the difference of the velocity pressures required by the two areas.

General Formulas for Abrupt Expansion.- The shock pressure loss occasioned by a faster moving stream expanding symmetrically and abruptly to a slower moving stream is given in all elementary hydraulics as equivalent to the velocity pressure corresponding to the difference of the two velocities involved. We have no such general law for unsymmetrical expansion and the following discussion will therefore apply only to symmetrical expansion unless otherwise stated. The shock loss for symmetrical expansion at any density, d, is therefore

$$H = 0.0^{6}831\, d(V_c - V_e)^2 \qquad (19)$$

and at standard air density of 0.075 pound per cubic foot, is

$$H = 0.0^{7}623\, (V_c - V_e)^2 \qquad (20)$$

where H is the pressure loss in inches of water
and V_c and V_e are the velocities in feet per minute at the smaller (contracted) and larger (expanded) areas respectively.

Where the actual velocities are known, the shock loss may be computed directly from this formula, but for general purposes it is desirable to have the loss in terms of equivalent velocity pressures, or shock factors, based on actual airway areas or ratios of airway areas.

If A_c and A_e represent the smaller and larger areas, respectively, involved in the expansion, then, since the quantity of flow is the same at each area, we have the relation $A_cV_c = A_eV_e$ from which may be obtained

$$(V_c - V_e)^2 = \left(\frac{A_e}{A_c} - 1\right)^2 V_e^2 \tag{21}$$

and

$$(V_c - V_e)^2 = \left(1 - \frac{A_c}{A_e}\right)^2 V_c^2 \tag{22}$$

and, with H_{ve} and H_{vc} representing the velocity pressures corresponding to velocities V_e and V_c, respectively,

$$H = \left(\frac{A_e}{A_c} - 1\right)^2 H_{ve} \tag{23}$$

and

$$H = \left(1 - \frac{A_c}{A_e}\right)^2 H_{vc} \tag{24}$$

Where the actual areas involved are known, as in simple expansion, these formulas can be used directly for computing the shock loss, or shock factors may be computed from the area ratios; but in the more general case of expansion following contraction, A_c is the area at the "vena contracta" and can not be measured directly but must be determined from experimental data. For this reason it is desirable to develop more general formulas that may be applied in all cases.

The general case of expansion following contraction is represented in Figure 6. Four areas are involved: A_a, the area preceding the contraction; A_o, the area at the contraction or orifice; A_c, the area occupied by the air flow at the "vena contracta;" and A_e, the area following the contraction, to which expansion occurs. If we designate the ratio of air-flow contraction at the constricted area by c, usually termed the "coefficient of contraction," and the ratio of expansion areas by N_e, then $\frac{A_c}{A_o} = c$, and $\frac{A_o}{A_e} = N_e$. Using these new designations for area ratios, we have

$$\frac{A_e}{A_c} = \frac{A_e}{c\ A_o} = \frac{1}{c\ N_e}, \tag{25}$$

and

$$\frac{A_c}{A_e} = \frac{c\ A_o}{A_e} = c\ N_e, \tag{26}$$

and, by substituting in (23) and (24),

$$H = \left(\frac{1}{c\,N_e} - 1\right)^2 H_{ve}, \tag{27}$$

and

$$H = (1 - c\,N_e)^2\,H_{vc}. \tag{28}$$

The latter form is not directly usable, since it involves an unknown velocity pressure, that at the "vena contracta." However, with quantity of flow constant, velocity pressures vary inversely as the square of the areas involved, or ratio of areas, and, with H_{vo} denoting the velocity pressure at the contracted area A_o,

$$H_{vc} = \left(\frac{1}{c}\right)^2 H_{vo}; \tag{29}$$

so, by substitution in (28),

$$H = (1 - c\,N_e)^2 \left(\frac{1}{c}\right)^2 H_{vo} = \left(\frac{1}{c} - N_e\right)^2 H_{vo}. \tag{30}$$

Although it is not directly involved in the pressure loss, it is often convenient to express the pressure loss in terms of the velocity pressure corresponding to the area, A_a, preceding the contraction. If we let N_c denote the ratio of the contracted area to the area preceding contraction, or $\frac{A_o}{A_a}$, and H_{va} the velocity pressure corresponding to the area A_a preceding contraction, then from the inverse relation of velocity pressures to the square of the area ratio we have,

$$H_{vo} = \frac{H_{va}}{N_c^2}, \tag{31}$$

and, by substituting this value of H_{vo} in (30),

$$H = \frac{\left(\frac{1}{c} - N_e\right)^2}{N_c^2} H_{va}. \tag{32}$$

The shock loss is seen to be a constant number of velocity pressures for constant conditions and, in terms of velocity pressures corresponding to three different airway areas involved, is

$$H = X_e\,H_{ve} = X_o\,H_{vo} = X_a\,H_{va}, \tag{33}$$

where X_e, X_o and X_a are the shock factors based on the area after expansion, the area at the constriction, and the area preceding contraction, or A_e, A_o and A_a, respectively, in Figure 6. Convenience alone usually dictates which factor should be used. From (27), (30), and (32) the shock factors are,

$$X_e = (\frac{1}{cN_e} - 1)^2, \quad (34)$$

$$X_o = (\frac{1}{c} - N_e)^2, \quad (35)$$

and

$$X_a = \frac{(\frac{1}{c} - N_e)^2}{N_c^2} \quad (36)$$

for general application to conditions where the shock loss is due to abrupt symmetrical expansion in the area actually occupied by the flow.

General Formula for Gradual Expansion.- Where the expansion of the area occupied by the air stream is gradual and symmetrical, it has been found that the shock loss is a constant ratio of that for abrupt symmetrical expansion; and the required shock factors are

$$X' = y\,X, \quad (37)$$

where X' represents the shock factors for gradual expansion corresponding to values of X for abrupt expansion, and y is an empirical factor based on experiments.

Values of y vary with the included angle, or slope, of the walls, and values of y, available from experiments, are plotted in Figure 7 against the included angle. The "Fan Engineering" handbook[9] data indicate that angles greater than 60° are no better than abrupt expansion. These data do not show, however, that the normal angle of symmetrical expansion of air flow is on about 7°, above and below which value the shock loss is increased. Brigg's[10] data do bring out this fact, which is proved by a wealth of corroborating experimental data on air flow, and also indicate that conditions of non-symmetrical expansion have an effect on the shock loss, since the minimum is at 11° instead of 7° for 2-sided expansion, and it is probable that tests for one-sided rectangular expansion would show a still higher value for the optimum angle of expansion. The values of y plotted in Figure 7 for Briggs' data on 2-sided expansion were derived through the formulas for symmetrical (4-sided) expansion and can therefore be used with them, although not with any great degree of certainty.

Shock losses for gradual expansion depend on N_e as well as the experimentally determined values of y, and N_e is directly related to the slope of the sides and the length in diameters of the diverging section. The relations of slope to included angle, and of N_e to length and included angle, are also plotted, for convenience, in Figure 7.

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9 Buffalo Forge Co., Fan Engineering: 2d ed., Buffalo, 1925, p. 68.
10 Briggs, Henry, and Williamson, J. N., An Experimental Study of Fan Evasees: Trans. Inst. Min. Eng., London, vol. 68, 1924-25, pp. 323-344.

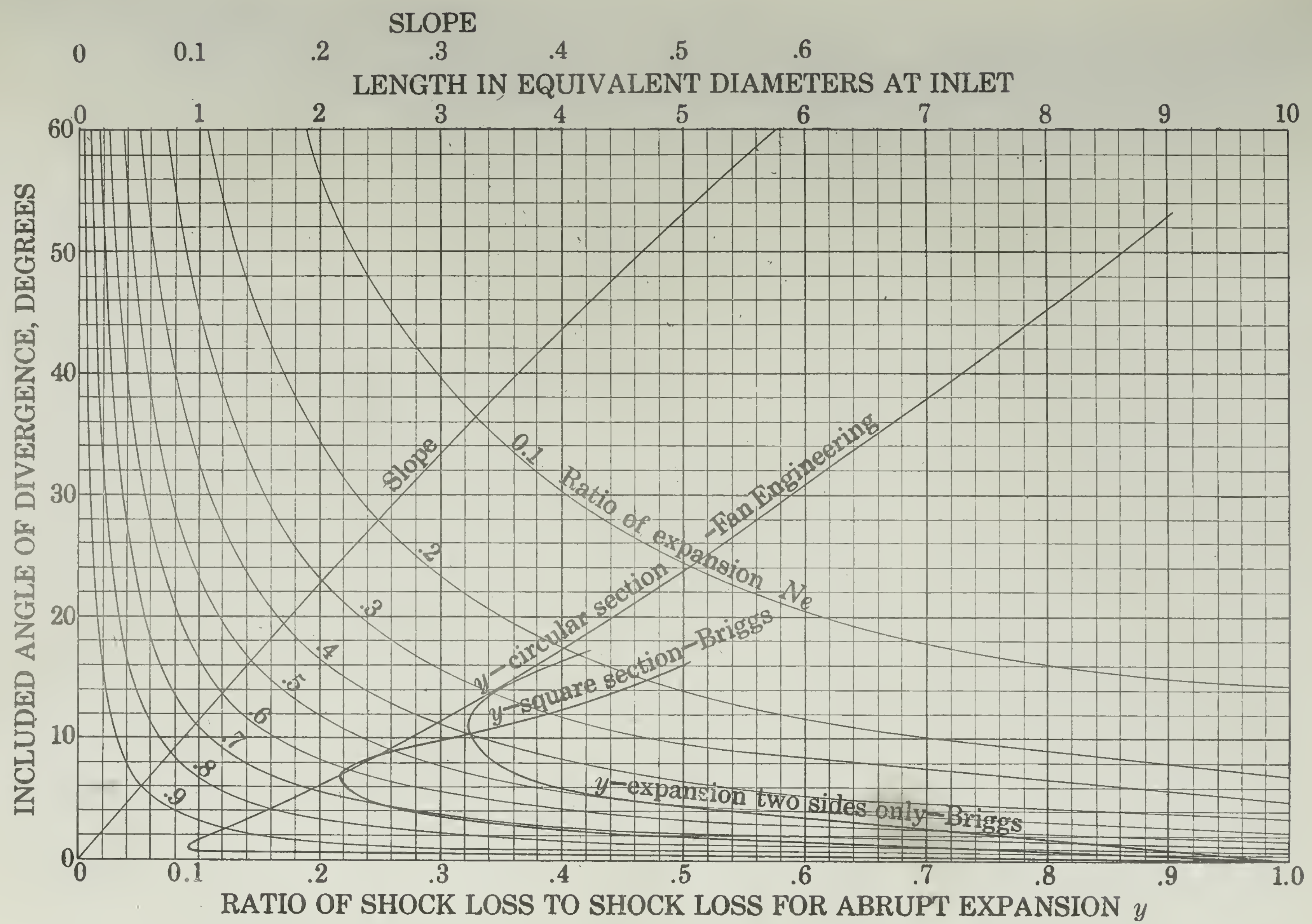

Figure 7.— Chart of data for gradual symmetrical expansion at diverging sections of airways, for included angles of 0° to 60°

Coefficient of Contraction.- Inspection of the formulas given for determining shock factors for symmetrical area changes shows that, in all cases of contraction, the coefficient of contraction, c, is involved. This coefficient, since it involves an area, A_c (fig. 6), that can not be measured, must always be calculated. As here used, it is a calculated figure determined by considering the total pressure loss at the condition, in excess of the normal friction loss, as a shock loss due entirely to symmetrical expansion, as it undoubtedly is, and computing the value of c accordingly. Values of the coefficient of contraction as given by various investigators do not always have this same meaning, as direct pressure measurements have sometimes led to a segregation of shock losses as occurring both before and after the "vena contracta." Those found preceding the constriction have always been so small that their actual existence is doubtful and, in any event, are included in the loss as determined by the coefficient of contraction as here used, which Weeks[11] designates as the "virtual" coefficient of contraction, and which others have termed the "coefficient of discharge."

Contraction Factors.- The author has found[12] by analyzing the results of his own tests and those of others, that, with other conditions constant, there is a definite relation between the ratio of contraction areas and the coefficient of contraction, which may be expressed thus:

$$c = \sqrt{\frac{1}{Z - ZN_c^2 + N_c^2}} \qquad (38)$$

where N_c is the ratio of contraction, or $\frac{A_o}{A_a}$ in Figure 6,

and Z is an empirical factor, which will be designated as the "contraction factor."

Values of Z calculated from test data, in which a series of coefficients are given for use in a particular formula, are usually practically identical for the middle range of values of N_c and depart from a practically uniform value only for the extreme ranges of N_c, in which ranges experimental accuracy is most difficult to obtain. For many of the conditions of contraction encountered, such as orifices in ducts of uniform area, the values of the ratios N_c and N_ϵ are identical, but such is not always the case, particularly as respects mine airways.

Values of Z are particularly affected by the edge condition at the constricted area, by the degree of abruptness of contraction, and by the form of contraction as regards symmetry. Practically all of the data available are for symmetrical contractions in which the areas of cross section are similar figures and centered as respects each other, and the data to be given are for such symmetrical layouts unless otherwise stated.

11 Weeks, W. S., The Air Current Regulator: Trans. Am. Inst. Min. Eng., vol. 76, 1928, p. 138.

12 McElroy, G. E., Ratio of Opening of Fan Performance: Jour. Am. Soc. Heat. and Vent. Eng., vol. 24, 1928, pp. 784-785.

Abrupt Symmetrical Contraction.- A large amount of data on abrupt and gradual contractions and expansions is given in the handbook "Fan Engineering,"[13] and includes coefficients and formulas for various conditions. Analysis of these and similar data indicates that there is very good agreement of the Z values for abrupt contraction obtained in various sets of tests, particularly when test results have been corrected for included friction pressure losses for several diameters of the duct used in the test installations. Minor differences are evident which may be sufficient to affect the accuracy of air measurements based on static pressure changes, but are not of sufficient magnitude to affect seriously the calculation of shock losses for mine airways.

For free contraction to a sharp edge, as at the entrance to a plain open pipe, Z averages 3.80. With a flat surface extending from the sharp edge at right angles to the direction of flow, a very common condition represented by mine airway entrances, by abrupt contraction in ducts and airways and by orifice plates, Z is about 2.5 for ordinary degrees of sharpness as obtained with square-edged plates, although the very sharp edges of thin orifice plates used for air measurement give a practically constant value of Z of 2.70. The exact edge condition has a large influence on the contraction factor and for the normal sharp-edges found in mine constructions, which will be referred to hereafter as "square" edges, a Z value of 2.5 is at least sufficiently high and generally applicable. If the edge is rounded, the shock loss is greatly reduced, but since the edge condition is not susceptible to rigid specification, this form is not applicable for air measurement, and there are therefore few data available for determining average Z values. For a well-rounded edge comparable to that presented by a round timber in a mine airway, the value of Z is estimated from scanty data as about 1.5, and for lack of a better-substantiated value, this will be used in determining shock factors for round-edged conditions in mine airways.

Although definite data are lacking, it seems probable that the same approximate Z factor of 1.5 can be applied where the edge is beveled to about the same degree - that is, to the extent that the bend would be circumscribed by a quadrant of a similar circular section. Most of the shock factors directly available apply to very sharp orifice-edge conditions, but no attempt will be made here to show shock factors for this type of edge condition, since we are concerned with pressure losses rather than air measurement.

Gradual Symmetrical Contraction.- If the contraction in area is made gradually instead of abruptly, the degree of contraction of the flow at the "vena contracta" is reduced and the shock loss also. Where the decrease in area is accomplished at a uniform rate, as in a well-rounded entrance or in the "standard" orifice, there is very little contraction of the flow and an average value of Z for the best designs is about 1.05. This form can be considered as representing a definite edge condition and will be referred to hereafter as a "formed" edge.

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13 See reference 9, pp. 60-72.

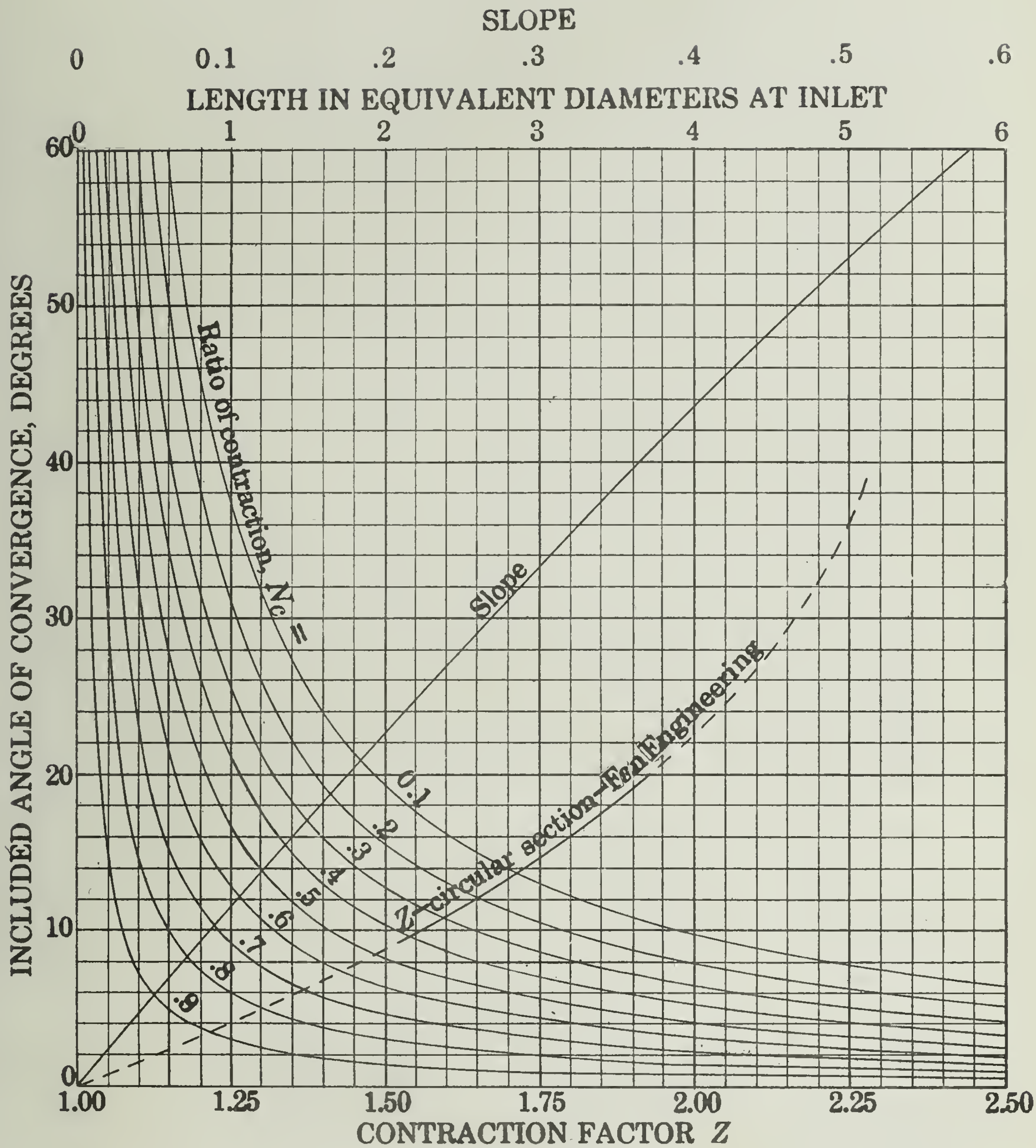

Figure 8.- Chart of data for gradual symmetrical contraction at converging sections of airways for included angles of 0° to 60°

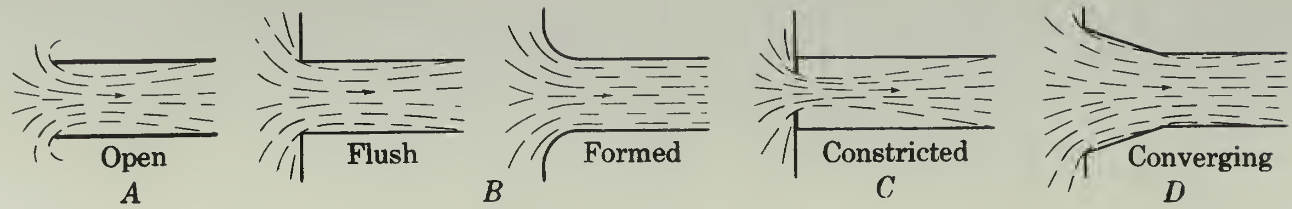

Figure 9.– Flow conditions at area changes at entrance to airways

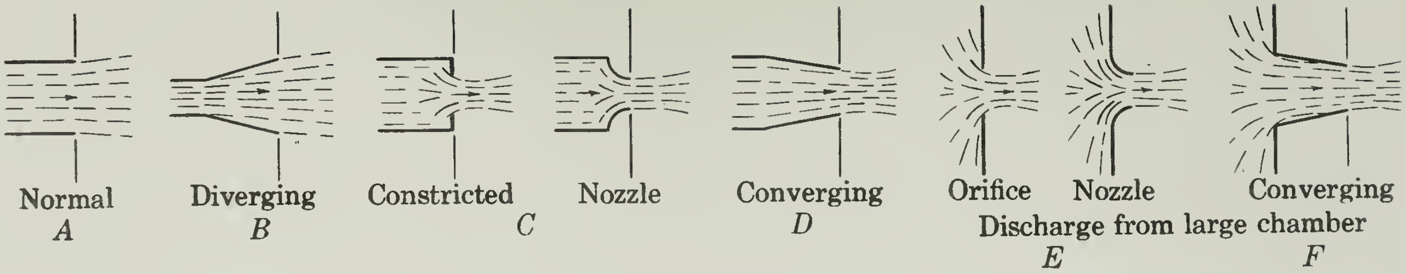

Figure 10.– Flow conditions at area changes at discharge from airways

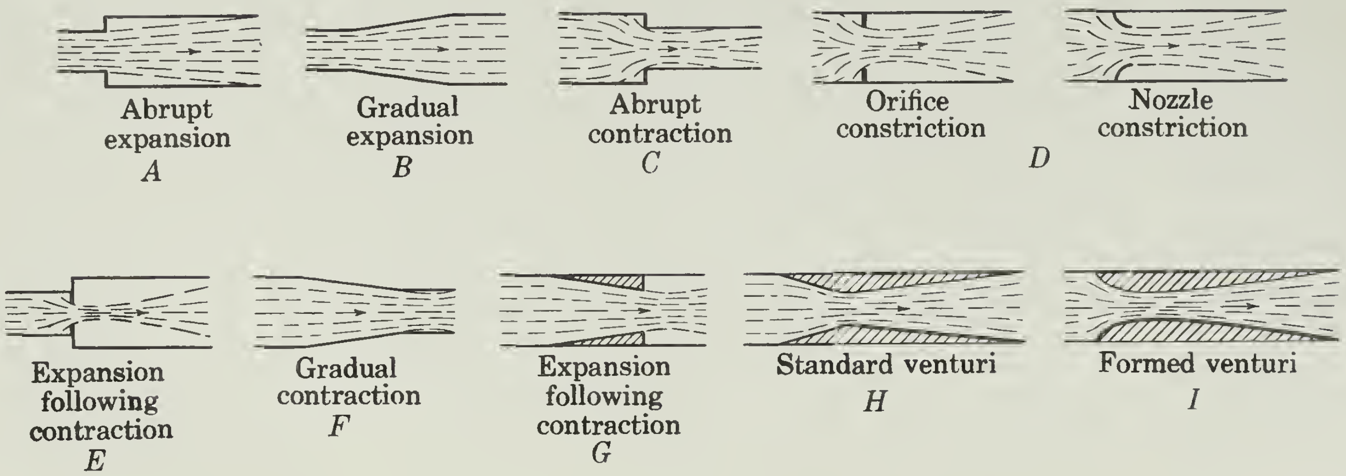

Figure 11.– Flow conditions at area changes within airways

For walls converging on a uniform slope, data are available only for the range of 10° to 20° included angle (Fan Engineering, p. 65) and assumed to be for a sharp edge. The calculated Z values are plotted against included angle in Figure 8, and range from 1.57 for 10° to 1.93 for 20°. At 0°, Z would be 1.0 and for abrupt contraction of 180° it would be about 2.5. The slope of the curve, however, indicates that the value of 2.5 is probably closely approximated somewhere between 45° and 60° and that greater degrees of included angle of convergence are no better, as regards shock loss, than abrupt contraction.

Shock Factors for Symmetrical Area Changes

Installation conditions may be considered in three groups for the purpose of presenting data on shock factors: (1) Entrance to an airway, Figure 9; (2) discharge from an airway to the atmosphere, Figure 10; and (3) within an airway, Figure 11. In discussing the various shock factors required, it is, however, more convenient to depart from this grouping and base the discussion on the conditions of expansion and contraction. Values of the shock factors corresponding to the generally applicable contraction factors of 2.5 for square edge, 1.5 for rounded edge, and 1.05 for a formed edge are tabulated in Table 1 for conditions such that either N_c or N_e have fixed values of 1 or 0, or $N_c = N_e$.

Simplified Formulas for Fixed Values of N_c and N_e.- Although the contraction factors for a particular type of contraction may be constant, the shock factors and pressure losses depend also on the actual values of the area ratios N_c and N_e involved in the installation. Their magnitude depends primarily on whether the flow expands abruptly or gradually or is contracted abruptly or gradually, but the range of required factors depends more particularly on whether N_c and N_e have variable or fixed values. These values are fixed to a large extent by the conditions of the installations, since the relation of the areas corresponding to A_a, A_o and A_e in the general case represented by Figure 6 may be such that either N_c or N_e may have fixed values of 1 or 0; and although the general formulas, (34) to (38), still apply, they reduce to simpler forms.

When $N_c = 1$, that is, when no contraction of the flow precedes expansion, Z and c are also 1, and the general forms reduce to

$$X_e = \left(\frac{1}{N_e} - 1\right)^2 \qquad (39)$$

and

$$X_a = X_o = (1 - N_e)^2. \qquad (40)$$

When $N_e = 1$, that is, when contraction is followed by a uniform area,

$$X_e = X_o = \left(\frac{1}{c} - 1\right)^2 \qquad (41)$$

and

$$X_a = \frac{\left(\frac{1}{c} - 1\right)^2}{N_c^2}. \qquad (42)$$

Table 1.- Shock factors, or shock pressure losses in terms of equivalent velocity pressures, for symmetrical changes of area of airways for common conditions of abrupt[1] contraction and expansion in which the ratios of contraction (N_c) or expansion (N_e) are 1, 0, or equal to each other

Constant ratio and special formulas	Variable ratio, of contraction (N_c), or expansion (N_e) — Condition of area change	Contraction factor, Z		1	.9	.8	.7	.6	.5	.4	.3	.2	.1	0
				Shock factor based on area before contraction (or expansion), X_a; on area at contraction, X_o; and on area following expansion (or contraction), X_e										
$N_c = 1$ $X_a=X_o=(1-N_e)^2$	Abrupt expansion without preceeding contraction	(1.0)	$X_a=X_o$	0	0.010	0.040	0.090	0.160	0.250	0.360	0.490	0.640	0.810	1.0
$X_e=(\frac{1}{N_e}-1)^2$			X_e	0	.012	.063	.184	.445	1.00	2.25	4.99	16.0	81.0	∞
$N_c = 0$	Wall contraction to													
$X_o=(\frac{1}{c}-N_e)^2$	Square edge	2.50	X_o	0.340	0.462	0.608	0.774	0.960	1.17	1.39	1.64	1.90	2.19	2.50
$X_e=\frac{(\frac{1}{c}-N_e)^2}{N_e^2}$			X_e	.340	.569	.953	1.58	2.67	4.66	8.70	18.2	47.6	219	∞
	Round edge	2/1.50	X_o	.050	.105	.180	.275	.389	.524	.679	.854	1.05	1.26	1.50
			X_e	.050	.130	.280	.561	1.08	2.10	4.24	9.49	26.2	126	∞
$X_a = ∞$ $c = \sqrt{\frac{1}{Z}}$	Formed edge	1.05	X_o	.001	.016	.051	.106	.181	.276	.391	.526	.681	.856	1.05
$N_e = 1$	Wall contraction to													
$X_e=X_o=(\frac{1}{c}-1)^2$	Square edge	2.50	X_a	0	0.022	0.091	0.220	0.446	0.839	1.59	3.21	7.92	33.2	∞
			$X_o=X_e$	0	.018	.058	.108	.161	.210	.254	.289	.317	.332	.338
$X_a=\frac{(\frac{1}{c}-1)^2}{N_c^2}$	Round edge	2/1.50	X_a	0	.003	.012	.030	.061	.120	.230	.471	1.18	4.96	∞
			$X_o=X_e$	0	.002	.007	.015	.022	.030	.037	.042	.047	.050	.050
	Formed edge	1.05	X_a	0				.001	.001	.003	.005	.015	.062	∞
			$X_o=X_e$	0								.001	.001	.001
	Obstruction with Square edge	2/5.00	X_a	0	.132	.495	1.13	2.19	4.00	7.41	14.8	36.0	151	∞
			$X_o=X_e$	0	.107	.317	.554	.787	1.00	1.19	1.33	1.44	1.51	1.53
	Round edge	2/3.00	X_a	0	.038	.152	.361	.723	1.35	2.54	5.14	12.5	52.7	∞
			$X_o=X_e$	0	.031	.097	.177	.260	.338	.406	.462	.501	.527	.536
	Formed edge	2/2.10	X_a	0	.012	.051	.127	.258	.503	.936	1.91	4.71	19.8	∞
			$X_o=X_e$	0	.010	.033	.062	.093	.126	.150	.171	.188	.198	.202

Table 1.- Shock factors, or shock pressure losses in terms of equivalent velocity pressures, for symmetrical changes of area of airways for common conditions of abrupt[1] contraction and expansion in which the ratios of contraction (N_c) or expansion (N_e) are 1, 0, or equal to each other - Continued

Constant ratio and speciall formulas	Condition of area change	Contraction factor, Z	Variable ratio, of contraction (N_c), or expansion (N_e)	1	.9	.8	.7	.6	.5	.4	.3	.2	.1	0
				Shock factor based on area before contraction (or expansion), X_a; on area at contraction, X_o; and on area following expansion (or contraction), X_e										
$N_e = 0$	Wall contraction to													
$X_o = \frac{1}{c^2}$	Square edge	2.50	X_a	1	1.59	2.41	3.60	5.45	8.50	14.1	26.3	61.1	249	∞
			X_o	1	1.29	1.54	1.77	1.96	2.13	2.26	2.36	2.44	2.49	2.50
$X_a = \frac{1}{c^2 N_c^2}$	Round edge	2/1.50	X_a	1	1.35	1.84	2.56	3.67	5.50	8.88	16.2	37.0	150	∞
			X_o	1	1.10	1.18	1.26	1.32	1.38	1.42	1.45	1.48	1.50	1.50
$X_e = \infty$	Formed edge	1.05	X_a	1	1.25	1.59	2.09	2.86	4.15	6.51	11.6	26.3	105	∞
			X_o	1	1.01	1.02	1.03	1.03	1.04	1.04	1.04	1.05	1.05	1.05
$N_e = N_c (=N)$	Wall contraction to													
$X_o = (\frac{1}{c} - N)^2$	Square edge	2.50	$X_a = X_e$	0	0.068	0.304	0.807	1.78	3.67	7.61	17.0	46.4	218	∞
			X_o	0	.055	.195	.395	.641	.918	1.21	1.53	1.86	2.18	2.50
$X_a = X_e = \frac{(\frac{1}{c} - N)^2}{N^2}$	Round edge	2/1.50	$X_a = X_e$	0	.027	.128	.361	.836	1.81	3.92	9.12	25.9	126	∞
			X_o	0	.021	.082	.177	.301	.453	.627	.820	1.03	1.26	1.50
	Formed edge	1.05	$X_a = X_e$	0	.014	.068	.199	.480	1.08	2.41	5.79	17.0	85.6	∞
			X_o	0	.011	.044	.098	.173	.269	.385	.521	.679	.856	1.05
	Obstruction with Square edge	2/5.00	$X_a = X_e$	0	.224	.903	2.23	4.60	9.00	17.8	38.2	100	452	∞
			X_o	0	.182	.582	1.09	1.66	2.25	2.85	3.44	4.00	4.52	5.00
	Round edge	2/3.00	$X_a = X_e$	0	.093	.409	1.06	2.30	4.68	9.57	21.1	56.9	264	∞
			X_o	0	.075	.262	.520	.828	1.17	1.53	1.90	2.27	2.64	3.00
	Formed edge	2/2.10	$X_a = X_e$	0	.049	.227	.616	1.38	2.92	6.09	14.4	38.1	181	∞
			X_o	0	.040	.146	.302	.497	.730	.974	1.30	1.52	1.81	2.10

1/For gradual contraction, interpolate in table for value of Z from Figure 8. For gradual expansion, with or without preceeding contraction, multiply tabular values by Y from Figure 7.
2/Tentative approximations only.

When $N_c = 0$, that is, when the area preceding contraction is so large that it may be considered to be infinite in comparison with the contracted area, the general form (38) for coefficient of contraction reduces to

$$c = \sqrt{\frac{1}{Z}}; \qquad (43)$$

that is, the coefficient is constant for fixed types of contraction.

When $N_e = 0$, that is, when the area following the contraction is so large that it may be considered to be infinite in comparison with the contracted area, the general forms reduce to,

$$X_0 = \left(\frac{1}{c}\right)^2 \qquad (44)$$

and

$$X_a = \left(\frac{1}{cN_c}\right)^2 . \qquad (45)$$

When both N_c and N_e are 0, as in discharge through a small hole in a large plate, both (43) and (44) apply and,

$$X_0 = Z . \qquad (46)$$

When $N_c = 1$ and $N_e = 0$, which is the condition for unrestricted free discharge from a duct or airway, (40), (44) and (45) all reduce to

$$X_a = X_0 = 1 . \qquad (47)$$

Abrupt Expansion.- Where no contraction of the flow precedes the abrupt expansion, we have what amounts to a special case of application of the general formulas which provide for such contraction, in which the value of N_c, Z and c is 1.0. There are only two common conditions of installation: In the airway (fig. 11-A) where N_e may have any value between 0 and 1, and formulas (39) and (40) apply; and at discharge into the atmosphere (fig. 10-A) where N_e is 0 and $X_a = 1$, from (47).

Gradual Expansion.- Where no contraction of the flow precedes gradual expansion, we also have what amounts to a special case of application of the general formulas which provide for such contraction, since the value of N_c, Z and c is 1.0; but the shock factor, from (37), is y times the factor for abrupt expansion as determined from (39) and (40), and y is derived from Figure 7, or a similar source.

The most common condition of installation is that within the airway (fig. 11-B). This form, the diverging connection or transformation, finds an important application in reducing pressure losses in mine airways by substituting gradual expansion for abrupt expansion. The same type of construction may be used at the discharge end of the airway (fig. 10-B), but gradual expansion is then followed by abrupt expansion, and the factors for this combined shock loss are different than for gradual expansion only, although they can be combined into single factors in terms of the normal airway area or the area at discharge. This form is also of great practical importance for reducing the pressure losses at the discharge from mine airway systems. As

applied to a fan discharging to the atmosphere, it is termed an "evasee" discharge, the main purpose of which is to reduce the velocity at the point where abrupt expansion occurs. When a fan with an evasee is connected to discharge into an airway of the same size, abrupt expansion does not occur at this point, but at the actual discharge from the airway system.

For practical purposes, it is rarely economical to use a length that will make the discharge area more than four times the area at entrance to the diverging section, as the reduction in pressure loss for N_e ratios less than 0.25 is almost insignificant.

Abrupt Contraction.- Abrupt contraction followed by abrupt expansion is a common cause of pressure loss in mine airways and embraces all three common conditions of installation. This is not readily apparent unless it is realized that the areas actually occupied by the flow rather than airway areas are the determining factors. The common picture of abrupt contraction is that where a smaller section follows a larger section with abrupt transition in areas, as in Figure 11-C. Because the smaller area is the same as the opening, or orifice, the fact that abrupt expansion - from "vena contracta" to the smaller airway area - follows abrupt contraction is somewhat obscured. Here $N_e = 1$, and (38), (41), and (42) apply. Unrestricted entrance conditions present a quite similar form (figs. 9-A and 9-B), if the area preceding contraction is considered as expanded indefinitely. In addition to the special condition of $N_e = 1$ we also have the special condition that $N_c = 0$, so that (41) and (43) apply. In Figure 9-A we have the special condition of free contraction to an edge rather than the common condition of contraction against, or guided by, a surface. The average value of X_e, as given by numerous test data, is 0.90 and represents a Z value of 3.8. The usual value given for the flush-surface entrance condition (fig. 9-B) which corresponds to the standard entrance form for mine airways, is $X_e = 0.47$. The corresponding value of Z is 2.84, indicating that this value is not only for a very sharp-edged condition but probably also includes some friction pressure loss in the test installation. A better value for mine airways is $X_e = 0.34$, based on Z = 2.5 for a square edge. With a round, or beveled, edge with a radius, or bevel, approximately equal to the wall thickness of a lined mine airway, a Z value of 1.5 is probably sufficiently high, and X_e would be reduced to 0.05, a very low figure to be so easily obtained.

A restricted entrance, as in Figure 9-C, has only the one special condition that $N_c = 0$, from which it follows that the coefficient of contraction, c, by (43), is 0.633 for a square edge (Z = 2.5), 0.817 for a round edge (Z = 1.5), and 0.976 for a formed edge (Z = 1.05).

Abrupt constrictions at discharge from a duct to the atmosphere (fig. 10-C) have only the special condition that $N_e = 0$, so that the special formulas (44) and (45) apply.

Flow through a small hole in the thin wall of a large chamber (fig. 10-E) may be considered as a special condition of constricted discharge. Both N_c and N_e are so small that they can be considered to be 0, and formula (46), $X_o = Z$, applies. The values of X_o for square edge, round edge, and formed edge are therefore 2.5, 1.5, and 1.05, respectively, for mine airway applications. The values usually given range from about 2.7 to 2.8 and are for orifice plates with very sharp edges.

A constriction in an airway of uniform area (fig. 11-D) has only the special condition that $N_c = N_e$, so that no subscript is required for N, but the general formulas (34), (35), (36), and (38) apply. However, the more usual condition of actual application of a constriction in mine airways will involve unequal areas and we have the general condition of abrupt contraction followed by abrupt expansion (fig. 11-E) for which the general formulas listed above are required. For both cases, either tables of values, or graphic charts, are desirable to obviate computations.

Gradual Contraction.- Assuming that gradual contraction is followed by abrupt expansion from the "vena contracta" to the area following contraction of the flow, the formulas and shock factors given for the preceding conditions of abrupt contraction would be affected only by the decrease in the normal value of Z, which may be determined roughly from Figure 8. For the special condition of contraction at a uniform rate against a formed surface, as represented by the "standard orifice" in common use for air measurement and pictured for particular conditions of installation in Figures 9-B, 10-C, 10-E, and 11-D, there seems to be general agreement on coefficients of contraction that give a value of Z = 1.05; in other words, there is very little contraction of the flow against a well-formed surface. This form is particularly useful at the entrance to a pipe, where a slight flare reduces the pressure loss tremendously, and where a well-formed "bellmouth" has the insignificant shock factor of $X_e = 0.0006$ as against 0.90 for an unflared sharp edge.

The more common form of gradual contraction is that where a section of the airway converges in the direction of flow, and makes a gradual transition of areas. The common picture is that of a larger section converging on a uniform slope of sides to a smaller section, as in Figure 11-F. Here N_e is 1 and formulas (38) with Z derived from Figure 8 or a similar source, and (41) apply.

In mine airways, a constriction that had been eased off on the approach side only might take the form shown in Figure 11-G, for which the general formulas are required, with Z values in (38) from Figure 8 or a similar source.

A converging discharge from an airway (fig. 10-D) has the special condition that $N_e = 0$, so that the special formulas (44) and (45) apply, with c from (38) using Z from Figure 8.

A converging section used as the discharge from a chamber (fig. 10-F) or as entrance to an airway (Figure 9-D) actually combines abrupt contraction with gradual contraction. The latter can be separately determined, but N_e is indeterminate for the abrupt expansion following the abrupt contraction condition at entrance. Test data in Fan Engineering, pages 64-65, for the first condition mentioned, indicate that the loss accompanying the abrupt contraction

at entrance is rapidly reduced as the included angle of the converging section is increased. Separate graphs are required for each condition of installation, but none are given here as these particular forms of installation of converging section have little practical application to mine airways.

Contraction Followed by Gradual Expansion.- Either abrupt or gradual contraction may be followed by gradual expansion, in which case the normal factors for abrupt expansion are multiplied by y, from Figure 7, according to (37). Few cases are met in practice except the combination of gradual contraction followed by gradual expansion. This combination allows of a large change of velocities and pressures with but a small pressure loss and is therefore extremely useful in fluid measurement methods based on pressure changes. The most common form is the standard "Venturi" (fig. 11-H), although equally good results may be obtained with the special "Venturi" (fig. 11-I) in which the converging section is replaced by the formed surface, for which Z may be taken as 1.05. Shock factors for the resulting shock loss, which primarily determines the cost of this form of air measurement for securing a continuous record, may be computed according to the general formulas (34) to (38) by the use of appropriate values of y from Figure 7 and, for the standard Venturi type, of Z from Figure 8, or similar sources.

Chart for Determining Shock Factors for Symmetrical Area Changes.- A graphic chart for the rapid determination of shock factors for the multitude of conditions of symmetrical contraction and expansion of the air stream that may be encountered in practice, is presented in Figure 12. It has four sections, marked A, B, C, and D, corresponding to the four possible steps in computations, all of which may or may not be required for a particular condition.

In A, the intersection of the proper value of Z with the curve for N_c determines the value of c from (38). In B, the intersection of the value of c with the curve for N_e determines the value of X_o from (35). Corresponding values of X_e and X_a are directly related to X_o through the velocity pressure-area ratio relations, $X_e = \frac{X_o}{N_e^2}$ and $X_a = \frac{X_o}{N_c^2}$; so in C the intersection of the proper value of X_o with the line for N_e determines the value of X_e, and, similarly, its intersection with the line for N_c determines the value of X_a. In D, the intersection of the proper value of X_o, X_e, or X_a for abrupt expansion with the line for the factor, y (from fig. 7), for the condition of gradual expansion, determines the corresponding value of X_o', X_e', or X_a' for gradual expansion. This section is not required, of course, for conditions of abrupt expansion; nor is section A required for conditions of abrupt expansion only, without preceding contraction, since we know that c = 1. In the latter case, $X_a = X_o$, and X_o for c = 1 is easily determined as the intercept of the N_e curve with the X_o scale in section B.

A continuous method of solving examples by the use of this chart is indicated on the chart by the solution of a particular example traced with arrows to show the directions of procedure. The example represents a condition of gradual expansion following abrupt contraction, with Z = 2.5, N_c = 0.25, N_e = 0.35, and y = 0.3. The solution shows that the shock factor corresponding

to the area after expansion, X'_e, is 3.5, and that corresponding to the area before contraction, X'_a, is 6.9. Ordinarily we would not be interested in X'_o, but it could also be determined through section D if desired. The intermediate values of c, X_o, X_e, and X_a are shown to be 0.645, 1.44, 11.8, and 23.1, respectively.

Shock Factors for Mine Airway Enlargements, Constrictions and Obstructions

Obstructions in mine airways, whether along the walls as in the more normal type of contraction, or out in the air stream, as in the case of a mine car or cage, form more or less abrupt contractions and expansions in the air stream and thus cause shock losses of the type under discussion. Some are symmetrical while others are not; contraction may occur on but one or more sides of rectangular openings, or only along the edges of obstructions in the airway; and an obstruction may be moving relative to the air flow and in the same or opposite direction. The few data that have been presented on symmetrical contractions and expansions, mainly on the few types involved in air measurement by pressure methods, do not begin to cover the field of shock losses in mine airways due to these causes and can serve only as a basis for very rough approximations until more data are obtained on the effect of unsymmetrical conditions of area changes on coefficients of contraction and shock loss.

Local Enlargements of Area and Dead Ends.- In mine airways there are many local enlargements of limited length in the area of the airway, and the change in area may be gradual or abrupt. Tests of a number of such conditions, such as shelter holes and breakthrough openings off coal-mine entries, dead-end stations off vertical metal-mine shafts, local widenings at side tracks in metal-mine drifts, and local increases in height in coal-mine entries at abandoned overcast locations, show that the increase over the normal for sections of airways containing such features is negligible. Such local enlargements usually occur on but one side of the airway and the abrupt unsymmetrical area changes are usually confined to dead ends of limited length in the direction of flow which the main current does not enter to any extent on account of its low angle of expansion. An eddy current will usually be found to be rotating in such dead ends, but the accompanying shock loss is negligible. Where the enlarged section is relatively long, the area gradations are usually less abrupt and the flow expands into the enlarged area. Shock losses occur due to gradual expansion and gradual contraction, but are counteracted by the decreased friction loss occasioned by the lower velocity of flow in the intermediate section, with the net result that the change in pressure loss over that of an equal length of normal cross-sectional area is usually negligible. The total pressure loss for sections including such enlargements may therefore be taken as equivalent to the friction loss for equal lengths of normal area.

Doorframes.- Doorframes at open doors on mine airways are a typical form of abrupt nonsymmetrical contraction, with contraction and expansion on but three sides of an off-center rectangle. Using the formulas for shock loss for symmetrical expansion as an approximation, computed Z values for two doors with square-timber frames, from Butte tests reported in Bulletin 261, pages 64-65, are 1.59 and 1.36. Since the exact edge conditions were not noted and

the values of N_c and N_e were somewhat indeterminate on account of very variable areas adjacent to the doors, these values are not considered very reliable. That the figures are lower than the standard of 2.5 for a square edge is no doubt partly due to gradual contraction on one side caused by the open door and partly to special edge conditions, which apparently more than offset the increase in shock losses between symmetrical and 3-sided expansion. Until better-substantiated methods are available, it is recommended that the formulas for symmetrical expansion be used with $Z = 2.5$ for a square-timber frame only, and $Z = 2.0$ for the same frame with the door open.

Regulators.- A common device used in the distribution of air currents in mines is a small opening in a stopping with the size regulated, usually by means of a sliding shutter, to cause a certain increased pressure loss on one flow circuit in order to improve the flow on other circuits. It is simply a device for causing a desired loss of pressure, and amounts to installing a square-edged orifice (fig. 11-D) in the airway. A value of 2.5 for Z is probably sufficiently high for a symmetrically placed regulator opening, but the value of Z that should be used for different degrees of unsymmetrical placing, particularly where the regulator is placed in a stopping that is parallel to the main flow, as at a split or junction, is entirely unknown at present.

Lagging Across Mine Airways.- The bureau's Butte test work gives values (Bull. 261, p. 54) of velocity pressure losses for lagging laid across the center of vertical shaft compartments, from which a Z factor of 5.2 has been computed, as for symmetrical expansion as before, for this type of contraction as caused by an obstruction to the air flow. For a center post of roughly squared timber in a coal-mine entry, the bureau's Experimental mine test work (Bull. 285, p. 51) gives data from which a Z value of 5.3 has been computed, which is in substantial agreement with the value for lagging. In contraction caused by obstructions of this type, contraction takes place only along the perimeter of the obstruction and not along the walls, and the high values of Z indicate that the degree of contraction, or the decrease in area from the edge to the "vena contracta," is much larger than for contraction across the whole perimeter at the contracted area. This effect is entirely rational, since we know that a greater width of the air flow is deflected for the same value of N_c, or ratio of contraction in area of the airway. For similar narrow obstructions, not centered either horizontally or vertically in the air stream, the values of Z would probably be less, since the velocities of flow are always a maximum at or near the center of the airway.

Mine Cars.- A mine car, or trip of cars, is a similar type of contraction of the air stream, but in this case the obstruction is so long that three pressure losses are involved. First, there is abrupt contraction, similar to the preceding case, except that the expansion following contraction takes place in the constricted area, somewhat similar to Figure 11-C, which is the same approximately as that of the orifice. Second, there is the increase in friction pressure loss due to flow for a certain length of airway in the constricted

section rather than in the full area of airway. And third, there is abrupt expansion from the constricted area to the full area of the airway, somewhat similar to Figure 11-A.

Calculation of the bureau's Experimental mine test data for pressure losses due to mine cars in coal-mine entry (Bull. 285, pp. 52-66), which are confirmed by a few tests in Butte metal mines, reported in Bulletin 261, page 135, gives a Z value, derived from the formulas for symmetrical area changes, for the front of a car (or cage) of about 5.35, which is quite similar to those calculated for lagging and center post, although an approximate value of 5 for all similar conditions is probably as close as the accuracy of the test data and the uncertainty of the method of computation justifies at present. This indicates that the Z values for obstructions are about twice those for wall contraction for similar edge conditions. The shock factors for obstructions given in Table 1 are based on this tentative assumption.

The increased friction loss due to increased velocities through the constricted area is rarely large enough to justify precise calculations and for ordinary purposes can be taken as equivalent to $\frac{2 - N}{N^2}$ times the normal loss for the same length of unobstructed airway, where N is the ratio of constricted area to normal area. The actual ratio of losses depends on the relative shapes of airway and obstruction and can, of course, be calculated exactly for the particular conditions. The expression given serves for computing the loss with an accuracy within about 5 per cent for average coal and metal mine airway conditions and is thus sufficiently precise.

Shielding Effect of Obstructions.- The bureau's experimental mine tests on cars also bring out an effect of closely-spaced obstructions to flow which is commonly referred to as the "shielding effect." When the obstructions to flow are so closely spaced that the flow does not have a sufficient distance downstream in which to expand fully before encountering another obstruction, the expansion is not carried out to the fullest extent possible and the shock loss is thereby diminished to that corresponding to a higher value of N_e than the actual areas would indicate. Tests on coal-mine cars in a coal-mine entry (Bull. 285, pp. 58-59) showed that the cars had to be spaced about 30 feet center to center (about 25 feet clear distance between cars) before the pressure loss due to a group of cars was equivalent to that for one car multiplied by the number of cars. With the cars in a closely-coupled train, the normal condition, there is but one front shock loss and one rear shock loss, and the intermediate cars contribute only the increased friction loss in the constricted airway proportional to the total length they occupy coupled together.

Moving Obstructions.- Although no test data appear to be available for cars, skips, and cages in motion, it is entirely possible that the pressure losses can be calculated approximately from the data on hand, provided that effective velocities - velocity of air movement plus or minus the velocity of movement of the obstruction - are used instead of actual velocities in

computing the shock and friction losses involved. However, a body in movement in an airway acts like a loose piston and generates pressure which may, or may not, be opposed to the normal direction of movement of the air current. Data on such pressure generation are undoubtedly available from aeronautical research, but no method of applying them to airway conditions has been developed, to the author's knowledge.

Large volume circulations are often temporarily reversed by rapid movement of skips and cages in shafts and trains of cars on horizontal airways, where the moving object makes a rather close fit with the airway or where the pressure inducing the circulation is small, as in natural-draft ventilation. In an open-timbered shaft the movement of skips or cages operating in balance has but a minor effect on the air flow, since opposing pressures are generated and there is always plenty of clearance for the air flow in adjoining compartments. Where the air flow is in separately lined compartments with small clearance, the piston effect is large but is usually limited considerably by the provision of openings between compartments at frequent intervals. As a rule, these are provided to facilitate shaft repairs but serve the further purpose of limiting the piston effect of cage and skip travel and thus reduce interference with the normal distribution.

Calculations based on observations of air velocities and traffic counts by bureau engineers in 1924 in the parallel Liberty vehicular tunnels in Pittsburgh, before the installation of mechanical ventilation and with ventilation entirely by natural draft and car-generated pressures, show net pressures generated per automobile of 0.005 to 0.02 inch of water per car for car speeds averaging about 30 to 35 miles per hour (2,640 to 3,080 feet per minute) and with an average cross section of car estimated to be about 23 square feet in tunnels of 468 square feet sectional area. The lower net pressures per car obtained at the higher velocities of flow, which ranged up to 1,000 feet per minute in the direction of car travel, and the higher pressures per car obtained with the lower velocities. As the velocity of flow increased, the difference in velocities between car travel and air travel decreased and there was also more shielding effect of one car on a closely following car, thus decreasing the net pressure generated per car. A considerable amount of test work will apparently be required before we will be able to gage even approximately the effects of moving obstructions in airways.

Shock Losses at Splits and Junctions

Where part of the air current leaves or enters the main current, we have splits or junctions which involve shock losses due to both changes in direction and area changes. The few test data available indicate that, for splits or leaving currents, the losses are comparable to bend losses except that the velocity pressure of the partial current, or current in the split, should be used. Where air currents enter the main stream, however, both reasoning and test data indicate that the normal deflection shock losses for the entering

stream are increased by its interference with the air stream of the main current. Exact quantitative data are lacking, but since the effects are known to be of considerable magnitude, it seems prudent to use X factors for normal conditions, based on the velocity pressure of the entering stream, multiplied by at least 1.5, or an allowance of 50 per cent.

It is evident that the shock losses at splits and junctions are largely determined by the proportional divisions of the total quantity of flow, which are dependent on the total of both friction and shock losses in the branch circuits as well as the shock losses at entrance and departure in each. Problems of distribution are therefore difficult. Approximate solutions require a reasonably close estimate of the proportional divisions of the flow, and accurate solutions are only arrived at through a process of trial and error.

Practically all splits and junctions in mine airways present the types of irregular conditions, such as nonuniform areas at bends and unsymmetrical area changes, for which we have no strictly rational methods of computation; and while we may have a few directly-determined shock factors for special combinations, we will be practically without data for these important types of pressure losses in mine airways until rational methods of computation have been developed.

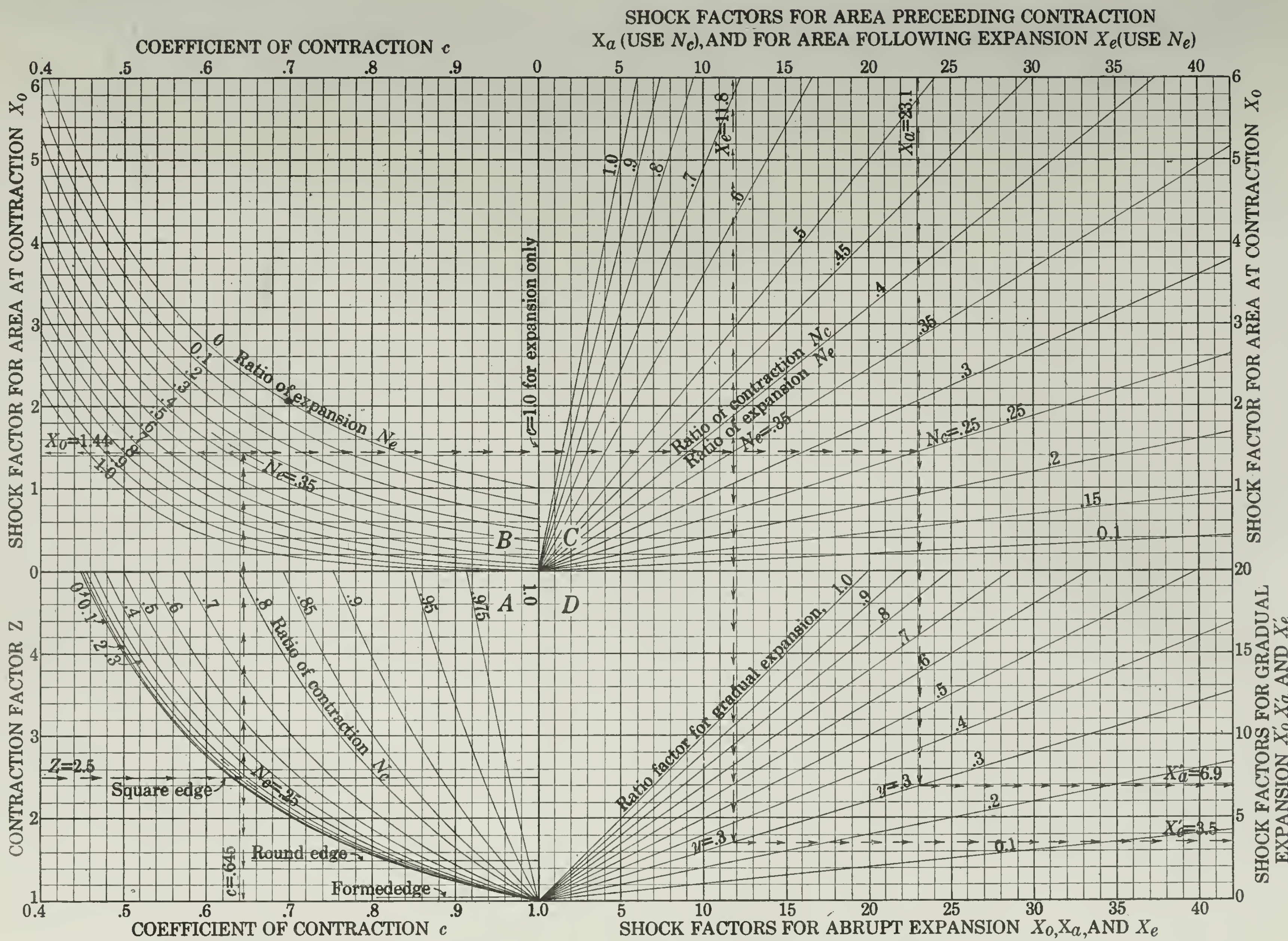

Figure 12.- Chart for determining shock factors, or shock pressure losses in terms of equivalent velocity pressures, for symmetrical changes in area of airways

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Scott Turner, Director

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ACCIDENT EXPERIENCE AND COST IN TENNESSEE COAL MINES

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

ACCIDENT EXPERIENCE AND COST IN TENNESSEE COAL MINES[1]

By F. E. Cash[2]

PURPOSE OF THIS REPORT

Much has been written concerning accidents in coal mines; the fact that accidents cost the employer and employee money is generally well known, but how much accidents actually cost is vague, even in the minds of many operators. Unquestionably there are hazards in coal mining, but many of them can be eliminated; and from the prevention of accidents there can be realized a material saving in dollars and cents as well as in human suffering.

In presenting the frequency, severity, and cost of accidents in some Tennessee coal mines, it is hoped that other mines in Tennessee, as well as mines in other States, will by a comparison of the data with their own experience be enabled eventually to reduce the number of their accidents.

TENNESSEE WORKMEN'S COMPENSATION LAW

The Tennessee Workmen's Compensation Law was approved on April 15, 1919, and amended on March 31, 1923, and April 14, 1927; it is entitled "An Act to Provide an Elective System of Workmen's Compensation for Industrial Accidents." This act covers coal mines employing five or more men.

A case of temporary disability compensation begins on the eighth day, and if it is of such nature that it extends over a period of 6 weeks or more, the first 7 days are compensable. The act further provides for payment of one-half the average weekly wage, stipulating a minimum of $5 and a maximum of $16 per week.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6664."

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Schedule of Compensation

The specified compensable periods are not given in full in the schedule of compensation, but they cover the accidents which are discussed in this report. In Table 1 the lost days allowed by the Tennessee compensation act appear in column 2; that the severity of accidents may be compared with experience in other mines and in other States, column 3 contains the lost days specified for permanent injuries used by the United States Bureau of Mines, the National Safety Council, and others in determining and comparing accident severity.

Table 1.- Scale of time losses for weighing deaths and permanent injuries in Tennessee

1	2	3
	Days lost	
Disability	Tennessee compensation law	National safety competition
Death	2,800	6,000
Permanent total disability	3,850	6,000
Loss of thumb	420	600
Loss of index finger	245	300
Loss of second finger	210	300
Loss of third finger	140	300
Loss of fourth finger	105	300
Loss of big toe	210	300
Loss of hand	1,050	3,000
Loss of arm	1,400	3,600
Loss of foot	875	2,400
Loss of leg	1,225	3,000
Loss of eye	700	1,800

TENNESSEE ACCIDENT EXPERIENCE

During the period 1926 to 1931, inclusive, the State of Tennessee produced 33,245,840 tons of coal with 142 fatal accidents in and around the mines. The fatality rate per million tons mined was below or better than the average for the United States for each year except 1926, when the State had its only major disaster, resulting in the death of 31 men, during the 6-year period.

That the accident experience may be comparable with coal mines of the State of Tennessee as a whole, it is deemed expedient to present in tabular form the relation of fatal accidents to operation of coal mines for the period 1926 to 1931, inclusive. Table 2 was compiled from publications and data furnished by the Tennessee Department of Mines.

Table 2.- Relation of fatalities to operation in all Tennessee coal mines, 1926-1931, inclusive, according to the State Department of Mines

Year	Mines operating	Coal produced, tons	Average days active	Total employed	Man-days worked	Fatal accidents	Tons per fatality	Man-days per fatality	Fatalities per million tons mined
1926	133	6,089,162	234	8,374	1,959,516	49	124,065	39,990	8.04
1927	111	5,935,150	221	7,574	1,673,854	17	349,126	92,579	2.86
1928	116	5,545,672	218	7,449	1,523,902	19	291,877	80,205	3.39
1929	81	5,663,155	241	7,720	1,860,520	21	269,676	88,596	3.68
1930	74	5,382,607	196	7,730	1,514,080	20	269,130	75,704	3.70
1931	71	4,630,094	185	6,794	1,256,890	16	289,381	78,556	3.48
Totals or averages	-	33,245,840	216	-	9,788,762	142	234,126	68,935	4.27

ACCIDENTS IN THIRTEEN REPRESENTATIVE MINES

The following data were furnished by representatives of 13 operating companies who requested that their names be withheld:

It has been possible to obtain the direct cost of accidents from 13 mines. By direct cost is meant money spent by the operating companies for medical attention, hospitalization, and compensation.

The mines throughout the tables are designated by numbers, beginning with 1 for the mine producing the most coal and 13 for the mine producing the least coal from 1926 to 1931, inclusive.

Seven of these mines operated for the entire 6-year period, 2 for four years, 2 for two years, and 2 for only one year. During the period for which figures were furnished the total production was 3,703,547 tons of coal with 772 accidents resulting in the loss of ~~109,666~~ 119,566 days.

These 13 mines cover those of various ages, sizes, and classes in Tennessee; among them are slope mines, drift mines just above water level and drift mines high up on the mountainsides with long surface inclines, new mines and old mines, large producers and small producers, mines abundantly equipped with machinery and mines in which practically all work is done by hand, and also gassy and so-called nongassy mines. Although there may be mines in the State with better or worse accident experience, these 13 mines may be considered representative.

Production and Accidents

In Table 3 the mines are numbered in order of tonnage produced. The production, years covered by these figures, average days worked per year, total man-days worked, total lost-time accidents, and the days lost are shown. In the "total accident" column are included all accidents causing the loss of one or more days other than that on which the accident occurred. In the "days lost" column are days allowed by the State compensation law for death or permanent disability; the actual days lost are given for accidents not causing permanent disabilities.

The total of 772 accidents included 21 fatalities, 4 total permanent disabilities, 42 partial permanent disabilities, and 705 other lost-time accidents.

Table 3.- Production and accidents at 13 mines in Tennessee 1926 to 1931, inclusive

Mine	Total coal production, tons	Years active	Average employed	Total man-days worked	Total accidents	Days lost
1	687,390	6	258.1	294,042	213	30,032
2	581,896	6	188.3	221,819	66	13,156
3	434,247	6	143.8	169,396	78	17,971
4	415,373	6	134.4	158,323	51	12,910
5	339,316	6	154.8	183,354	111	7,903
6	337,237	6	109.1	128,520	37	1,808
7	276,959	4	137.2	115,248	91	15,290
8	272,520	6	82.1	97,714	45	2,146
9	206,293	4	113.4	95,256	50	11,412
10	67,390	2	203.5	77,534	15	943
11	45,395	2	106.0	40,986	7	3,079
12	32,986	1	52.0	9,620	4	2,855
13	6,545	1	55.0	10,175	4	61
Total	3,703,547	-	-	-	772	-

Accident Frequency and Severity

Table 4 lists the accidents for every mine as to severity and frequency with the days lost, as furnished by the operating company, and the days lost according to the schedule of lost days used in the national safety competition conducted annually by the United States Bureau of Mines.

Frequency is the number of lost-time accidents multiplied by one million divided by the number of man-hours worked.

Severity is the days of time lost as a result of accidents multiplied by one thousand divided by the man-hours worked. In calculating the severity, the days lost are taken according to the national safety competition schedule.

Severity of accidents is a comparison of days lost as against days worked. In Table 4 the mines are arranged in the order of their severity rating; mine 13 has the best experience and mine 12 has the poorest; both of these mines operated only one year. A more nearly representative comparison would be in connection with mines working four or more years; on this basis mine 6 has the best severity experience and mine 7 the poorest. On a frequency basis, mine 7 has the poorest and mine 11 has the best record.

Table 4.- Accidents classified according to severity and frequency in 13 Tennessee mines, 1926 to 1931, inclusive

Mine	Fatals	Total permanent	Partial permanent	Other lost time	Tennessee, days lost	National competition, days lost	Total man-hours worked	Frequency	Severity
13	0	0	0	4	61	61	81,400	49.14	0.75
10	0	0	0	15	943	943	620,272	24.18	1.52
6	0	0	1	36	1,808	1,793	1,028,160	35.98	1.74
8	0	0	3	42	2,146	4,143	781,712	57.56	5.29
5	1	0	6	104	7,903	13,063	1,466,832	75.67	8.90
2	2	0	11	53	13,156	23,954	1,772,152	37.24	13.51
11	1	0	0	6	3,079	6,279	327,888	21.34	19.12
4	4	0	1	46	12,910	27,786	1,266,584	40.26	21.93
1	4	2	8	199	30,032	52,789	2,352,332	90.53	22.44
3	3	1	5	69	[illegible]	31,538	1,355,168	57.57	23.27
9	2	1	3	44	11,412	22,152	762,048	65.48	29.07
7	3	0	4	84	15,290	28,437	921,984	98.70	30.84
12	1	0	0	3	2,855	6,055	76,960	51.98	78.67

Cause of Accidents

In the classification of accidents, falls of roof and coal are responsible for the most lost days and haulage caused the greatest number of accidents. Flying coal caused many eye injuries, and employees on mining machines, in proportion to the number employed, have a high accident frequency. In the surface accidents, haulage has the highest frequency and severity. These accidents are listed by causes in Table 5.

Table 5.- Causes of accidents in 13 Tennessee mines, 1926 to 1931, inclusive

Cause	Fatal	Total dis-ability	Part dis-ability	Other lost-time accidents	Total lost-time accidents
Underground:					
Falls of roof or coal..	12	4	11	191	218
Flying coal or rock ...	0	0	5	41	46
Haulage	1	0	16	228	245
Electricity	2	0	0	13	15
Mining machines	0	0	3	29	32
Gas ignitions	0	0	1	3	4
Machinery	0	0	0	6	6
Explosives	0	0	0	5	5
Handling material	0	0	0	64	64
Tools	0	0	2	30	32
Falls of person	0	0	0	20	20
Nails	0	0	0	13	13
Miscellaneous	0	0	1	10	11
Surface:					
Haulage	6	0	1	14	21
Railroad cars	0	0	2	4	6
Machinery	0	0	0	10	10
Handling materials	0	0	0	15	15
Falls of person	0	0	0	5	5
Tools	0	0	0	4	4

Cost of Accidents

According to H. W. Heinrich of the Travelers Insurance Co. there are two kinds of accident costs - direct and indirect; the direct cost comprises medical, hospitalization, and compensation charges; the indirect or hidden cost includes time lost by the injured for which payment is not made by law or time lost by other employees, injury to equipment, loss of production, overhead expense, and other incidental costs of similar nature which usually accompany an accident. It is estimated by Heinrich as a result of analysis of a large number of industrial accidents that the indirect or hidden cost is four times as great as the direct cost of accidents.

In Table 6 the direct cost of accidents for the 13 mines is given on a cost per ton basis; this cost varies from 2.66 to 8.15 cents, with an average of 5.21 cents. The direct cost of accidents over the 6-year period was $192,904. If the direct and indirect cost ratio of 4 to 1 is used, then the total cost of accidents amounts to $964,520, or 26 cents per ton of coal.

Assuming that these mines are representative of all Tennessee coal mines for the same period, the direct cost of accidents for the State would be $1,560,611 and the total cost $7,803,056, or annually a direct cost of $260,102 and a total cost of $1,300,509.

Table 6.- Accident cost at 13 Tennessee mines

Mine	Coal produced, tons	Medical-hospital administration	Compensation	Total direct cost	Cost per ton, cents
6	337,237	4,924	4,357	9,281	2.75
10	67,390	389	1,536	1,925	2.86
8	272,520	3,302	3,944	7,246	2.66
5	339,316	3,787	8,199	11,986	3.53
13	6,545	302	34	336	5.15
2	581,896	8,496	22,494	30,990	5.33
9	206,293	2,079	9,903	11,982	5.81
1	687,390	7,581	32,627	40,208	5.85
7	276,959	1,590	14,677	16,267	5.87
4	415,373	6,064	20,654	26,718	6.43
3	434,247	7,701	22,163	29,864	6.88
11	45,395	258	3,154	3,412	7.52
12	32,986	344	2,345	2,689	8.15

SUMMARY

Table 7 is a brief summary of the accident experience of the 13 mines. While this is not the best nor is it the worst accident experience for mines, groups of mines, or mining States, it will present an opportunity for those interested to compare their mines, districts, or State with some actual direct accident-cost figures.

Table 7.- Summary of accident data for 13 Tennessee mines in the period 1926-1931

Number of mines	13
Coal produced .... tons	3,703,547
Man-days worked	1,601,987
Fatal accidents	21
Total permanent disabilities	4
Part permanent disabilities	42
Temporary disabilities	706
Total lost-time accidents	772
Total time lost .... days	218,993
Time lost per accident .... days	283.7
Accidents per fatality	36.8
Coal per accident .... tons	4,797
Coal per day lost .... tons	16.9
Direct cost .... cents per ton	5.21
Frequency	60.25
Severity	17.09

CONCLUSION

The accidents in these 13 mines, as well as those in other Tennessee mines, unquestionably can be reduced in frequency, severity, and cost by the proper training of all employees and by better and closer supervision. Even if the total cost of accidents in the coal mines of Tennessee should not actually reach the 26 cents per ton as is indicated in the analysis in this paper, there is no question that the cost of coal-mine accidents not only in human suffering and misery but also in dollars and cents is excessive. There is also no question that all of these various kinds of accident costs can be materially decreased by the use of readily available accident prevention methods, and there is no good reason why at least half of present accident costs of all kinds in Tennessee coal mines can not be saved.

INFORMATION CORCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

THE SIGNIFICANCE OF THE BUREAU OF MINES APPROVAL OF GAS MASKS[1]

The approval of the U. S. Bureau of Mines when granted to a manufacturer for a gas mask is a certification that the particular device has been submitted to the Bureau of Mines for inspection and test and has been found to meet that bureau's published minimum requirements for safety, durability, and satisfactory performance.

Gas masks are approved by the Bureau of Mines only as a complete unit or assembly of parts, as canister, facepiece and harness. This is in order that responsibility for rigid adherence to the bureau's requirements may be definitely located and not divided. Any device marketed under this approval must be a unit composed of parts identical in detail of design, assembly, workmanship, and quality, and produced under the same control and inspection as the original device to which approval has been granted. The particular parts which comprise the approved assembly are designated in a letter to the manufacturer at the time the approval is granted. In recent years they are also specified in the approval plate authorized for use in connection with the sale of the device. Each part permitted in the assembly is legibly marked by name or number for the guidance of users of masks in ordering and identifying approved parts when making repairs, and thereby maintaining their equipment as an approved device.

In view of the foregoing, any change in the design of parts or deviation from the permitted assembly of parts would violate the conditions under which approval was granted and the device would not be considered as meeting the approval requirements of the Bureau of Mines. The U. S. Bureau of Mines has been frequently requested to interpret the approval status of a gas mask assembled from parts which have separately met its requirements as components of two or more approved devices but which in the form of a complete unit has not been submitted for tests or been granted approval. Responsibility for the elements and for their proper assembly is in such case divided. Such assembled units have no status as permissible equipment under the Bureau of Mines system of schedules. The reasons underlying this policy are that the approval is granted only to some one who assumes responsibility for the whole and that the details of assembly and coordination of parts are considered to be important to the satisfactory performance and safety of the device.

The requirements and tests are described in the following publications:

Schedule 14B, Procedure for Testing Gas Masks for Permissibility.
Schedule 19, Procedure for Testing Hose Masks for Permissibility.

For the convenience of purchasers and users of gas masks the Bureau of Mines issues currently a list of the devices which have met its requirements and to which approval has been granted. These publications may be obtained from the U. S. Bureau of Mines, Washington, D. c.

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Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

MINING METHODS AND COSTS AT THE NEW CORNELIA BRANCH, PHELPS DODGE CORPORATION, AJO, ARIZ.

BY

George R. Ingham and Alfred T. Barr

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

MINING METHODS AND COSTS AT THE NEW CORNELIA BRANCH, PHELPS DODGE CORPORATION, AJO, ARIZ.[1]

By George R. Ingham[2] and Alfred T. Barr[3]

INTRODUCTION

This paper is one of a series being prepared by the Bureau of Mines on mining methods and costs at various mines in the United States, and is a description of the mining practice at Ajo, Ariz., where the New Cornelia Branch of the Phelps Dodge Corporation mines copper ore from an open pit, and operates a concentrator for the treatment of the ore thus mined.

ACKNOWLEDGMENTS

The authors wish to express their appreciation to M. Curley, manager of the New Cornelia Branch of the Phelps Dodge Corporation, for his review of this paper and permission to publish it. They also wish to acknowledge the assistance given by W. C. Lawson, assistant engineer, in the preparation of the paper.

HISTORY

The copper deposit at Ajo was one of the first to be discovered in the Southwest, but its development into a successful mining enterprise occurred at a comparatively recent date. For a period of nearly a half-century after the discovery of copper in the district, mining was confined to small bodies of high-grade ore which were found on the edge of what was later proved to be a large deposit of low-grade copper ore. In the sixties, native copper and cuprite ore were mined from shallow surface workings and hauled by teams to San Diego and later to Yuma, and from these ports transported in sailing ships to Swansea, Wales. After the Southern Pacific Railroad was built, ore was hauled 43 miles to Gila Bend, which was the nearest shipping point on the railroad. The mining of small high-grade orebodies continued in an intermittent way until the panic of 1907, when all properties were closed down, and the camp was practically deserted. It is probable that at no time were these operations very profitable, the chief factors against their success being the lack of water in the district, the long haul to a seaport or railroad, and the high cost of mining the small bodies of ore. During this period practically no work had been done on the great carbonate outcrop of the main orebody; several years elapsed after the suspension of operations in 1907 before the potential worth of the deposit was realized.

The first real attempt to explore the orebody was made in the winter of 1909-1910. At that time the major portion of the mineralized area was held by two companies, the New Cornelia Copper Co. and the Rendall Ore Reduction Co. The Lewisohn interests secured an option

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6666."

2 - One of the consulting engineers, U. S. Bureau of Mines, and mine superintendent, New Cornelia Branch, Phelps Dodge Corporation.

3 - One of the consulting engineers, U. S. Bureau of Mines, and chief engineer, New Cornelia Branch, Phelps Dodge Corp.

on the New Cornelia property and put down five diamond-drill holes around the base of Copper Mountain, the most prominent outcrop in the basin. Although three of these holes cut material which is now considered ore of commercial grade, it was not considered so at that time, and the option on the property was given up. Likewise, Seeley W. Mudd and associates secured an option on the Rendall property, along the southern and eastern edges of the main orebody, but after drilling six holes, they gave up the option. Four of the holes were blanks and two cut ore averaging about 1-1/4 per cent copper.

Despite the disappointing results of these first attempts at drilling the deposit, the Calumet & Arizona Mining Co. took an option on all available stock of the New Cornelia Copper Co., and in December, 1911, started a drilling campaign, which eventually developed a considerable tonnage of low-grade ore. The first hole was drilled on the west side of Copper Mountain, 200 feet north of the best hole drilled by the Lewisohn company, and over 200 feet of ore was cut which averaged about 1-1/2 per cent copper. As the results of this hole were encouraging, drilling was continued until an area of nearly 80 acres had been explored, and 40,000,000 tons of ore averaging 1-1/2 per cent copper had been developed.

In the meantime, the Rendall property had been acquired and developed by James Phillips, jr., and James P. Gaskill, under the name of the Ajo Consolidated Copper Co. The ground lying between the Rendall and the New Cornelia properties, known as the Childs group, was taken under option by representatives of the U. S. Smelting, Refining & Mining Co., but was given up after a few churn-drill holes had been put down. The New Cornelia Copper Co. afterwards acquired the holdings of the Ajo Consolidated Copper Co., the Childs group of claims, and a large block of ground north of the original holdings, on which the townsite, tailings dumps, and plants are now located.

The orebody which was being developed at Ajo was different in many respects from other so-called "porphyry coppers" in the Southwest, and it was evident that some departures from the usual methods of mining and treatment would be necessary to insure the success of the enterprise. The upper part of the orebody consisted of oxidized or carbonate ore, which extended from the surface down to an average depth of more than 50 feet. This ore had about the same copper content as the sulphides, which lay immediately underneath, but at that time no process for the treatment of such low-grade oxidized ore was in use. As it seemed that economical mining of the orebody meant the use of open-pit rather than underground methods, a successful method of treating the carbonate ore, which had to be mined to uncover the sulphide ore, was necessary in order to bring the property to an early stage of production and to lessen the expense of stripping the sulphide ore.

Soon after development of the orebody was under way, experiments were being made to determine a method of treating the carbonate ore. It was found that the ore could be successfully leached with sulphuric acid, and the copper which went into solution could be deposited on sheets by electrolysis. After three and a half years of experimental work, during which about 15,000 tons of carbonate ore was treated in pilot plants, a commercial plant with a capacity of 5,000 tons daily was built. This plant was put into operation in May, 1917, and continued in operation until July, 1930, when practically all of the carbonate ore had been mined.

By 1923 sufficient sulphide ore had been uncovered to warrant building a concentrating plant for its treatment. The original plant, which had a rated capacity of 5,000 tons daily, was put in operation in January, 1924. A year or so before the carbonate ore was exhausted, the concentrator was enlarged and improved to give a capacity of approximately 15,000 tons daily.

Before construction work on the leaching plant was started, the Tucson, Cornelia & Gila Bend Railroad had been completed from Gila Bend to Ajo, and an adequate water supply for the subsequent operations had been developed in the valley 7 miles north of Ajo.

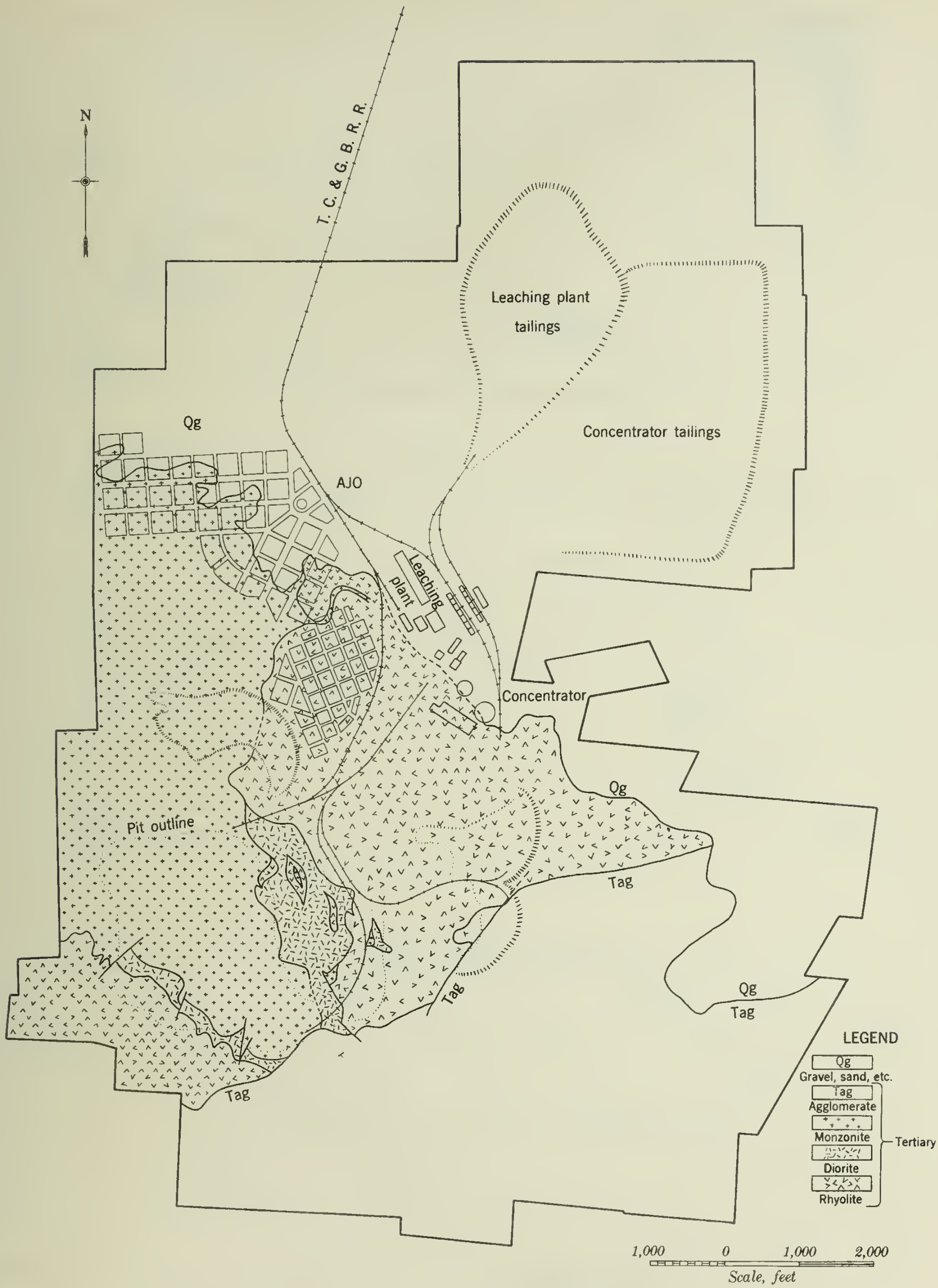

Figure 1.—Geologic map of the New Cornelia property

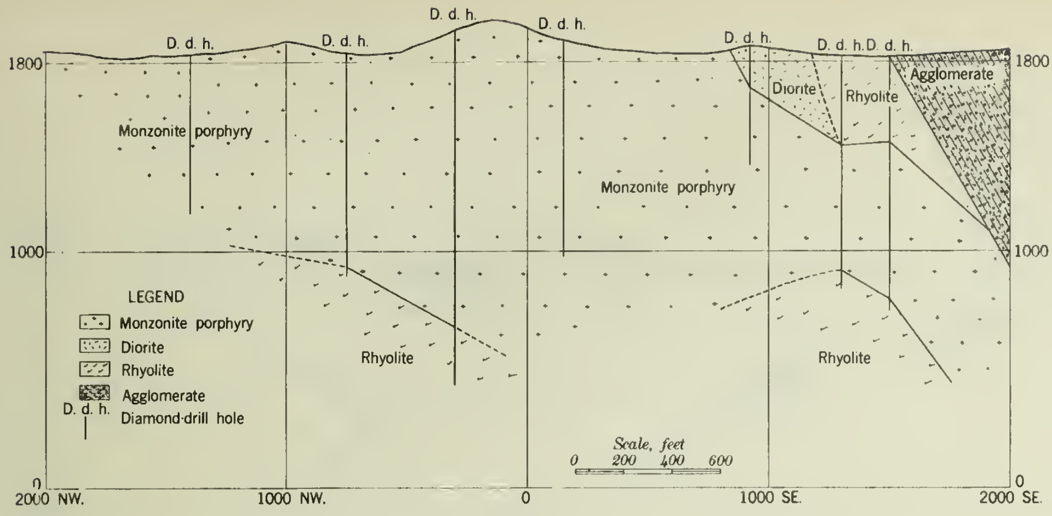

Figure 2.—Geologic section through orebody, looking northeast

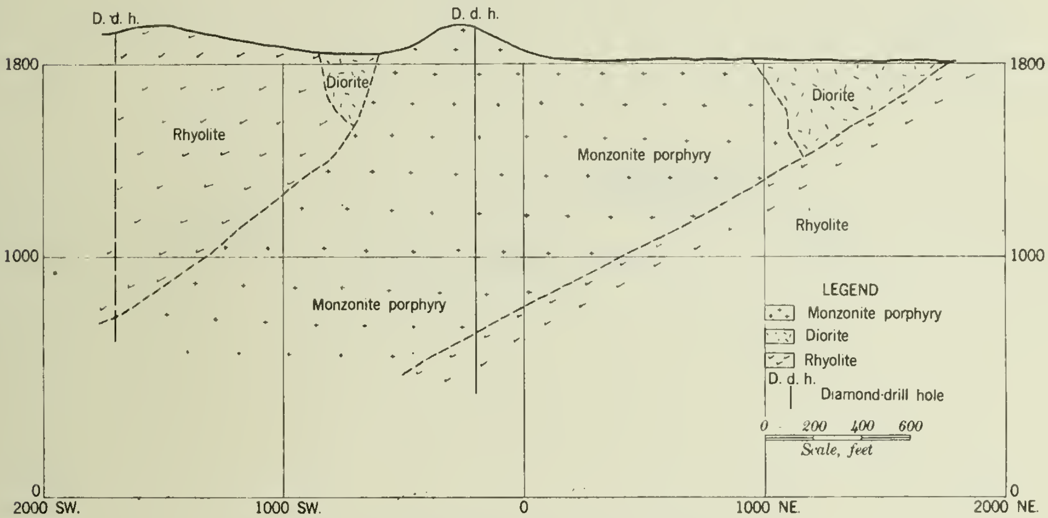

Figure 3.—Geologic section through orebody, looking northwest

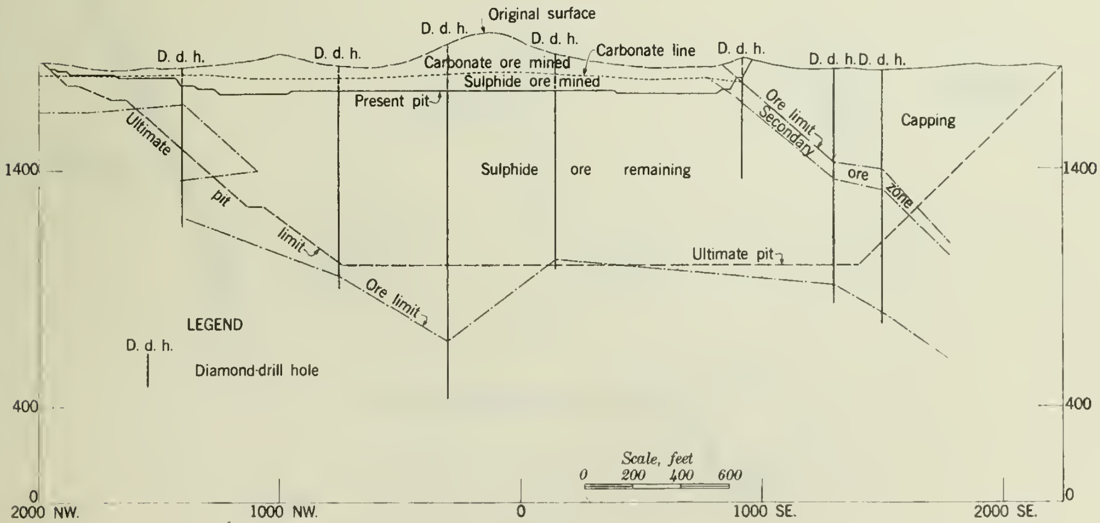

Figure 4.—Typical section of orebody, looking northeast

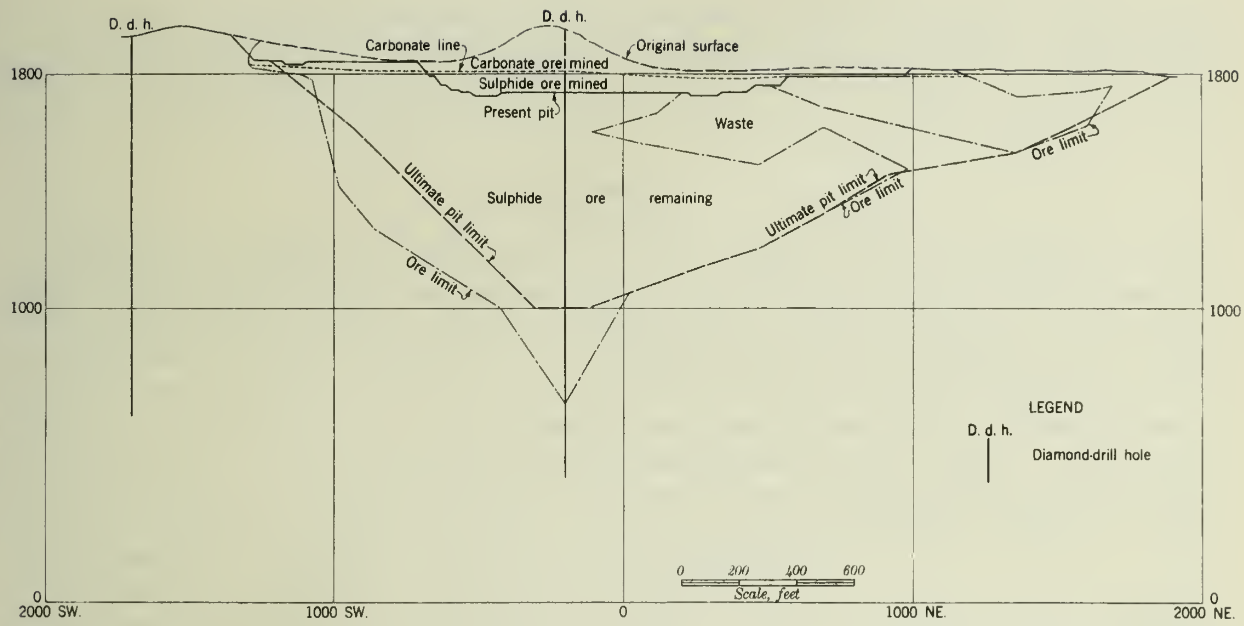

Figure 5.—Typical section looking northwest

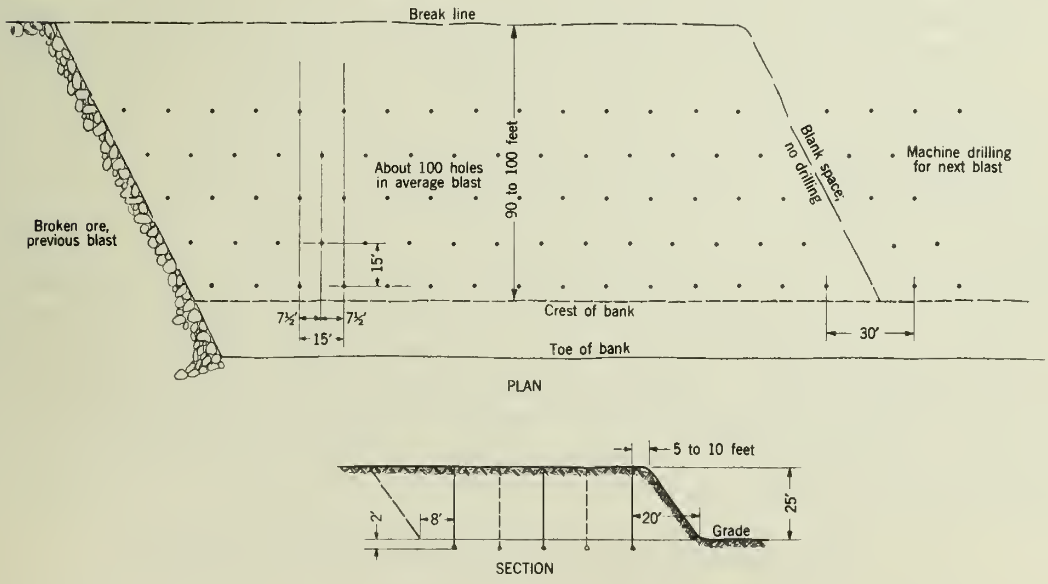

Figure 6.—Sketch of standard blast

In 1929 the New Cornelia Copper Co. was consolidated with the Calumet & Arizona Mining Co., and in 1931 this company consolidated with the Phelps Dodge Corporation. Since October 1, 1931, the New Cornelia property has been operated under the name of the New Cornelia Branch of the Phelps Dodge Corporation.

GEOLOGY

The orebody is located in the southern end of a large laccolith of monzonite and monzonite porphyry (fig. 1) which has intruded andesite and rhyolite flows, and which has, according to Joralemon,[4] uplifted the rhyolite beds to form a dome; the crest of the dome has been eroded away in fairly recent times. The exposed portion of the monzonite has a length of about 2 miles, and a width of over a mile at its northern end and about 1,000 feet at its southern end. The southern tip of the monzonite dips steeply toward the south and southeast, and is covered with rhyolite and beds of recent agglomerate (figs. 2 and 3). The northern end of the intrusive is covered with alluvium and recent lava flows. Bordering the southern point of the monzonite intrusion, there is a strip of diorite from 200 to 1,000 feet wide, which is presumably an earlier intrusion than the monzonite.

The orebody is composed of monzonite and diorite, and to a lesser extent rhyolite, which have been mineralized by copper-bearing solutions. Copper minerals have been deposited along fracture planes, and disseminated throughout the rock mass to form a large body of low-grade copper ore. During the primary mineralization, chalcopyrite, bornite, and some pyrite were deposited; then, after the capping of the orebody was eroded away and the top of the ore was exposed to oxidizing agencies, a blanket of oxidized ore was formed covering the primary ore. Secondary enrichment played only a minor part in the formation of the greater portion of the orebody, with the result that oxidized ore of nearly the same grade as the underlying primary sulphide ore was formed. Some secondary sulphide enrichment took place along the southeastern edge of the orebody, and here the primary ore is capped with chalcocite ore which is somewhat higher in copper than the main body of the oxide and sulphide ore.

The outcropping of the orebody is roughly circular in shape with a diameter of about 2/3 mile. The average thickness of the orebody, including ore which has been mined, is 425 feet, and the maximum thickness nearly 1,000 feet (fig. 4). Oxidized ore extended from the surface down to an average depth of 55 feet, and the depth varied from 20 to 190 feet; the contact between the oxidized ore and the underlying sulphide was, broadly speaking, nearly a level plane, but local variations of as much as 50 feet were encountered. The principal copper minerals in the oxidized ore are malachite and azurite, the former being the more prevalent. Chalcopyrite and bornite are the principal ore minerals in the primary sulphide ore, a greater proportion of the copper being in the form of chalcopyrite. The secondary ore zone forms a crescent about the southern edge of the orebody, and continues down on the flank of the monzonite under barren rhyolite and agglomerate. This secondary ore averages about 50 feet in thickness; chalcocite, in veinlets and finely disseminated throughout the rock, is the principal copper mineral. Other ore minerals which are found in the orebody, but not abundantly, are tetrahedrite, native copper, cuprite, tenorite, and chrysocolla.

PHYSICAL CHARACTERISTICS OF ORE AND ENCLOSING ROCKS

A typical cross section of the orebody (fig. 5) shows the sides sloping downward toward a definite keel of deep ore along the bottom. This keel, which is along the longitudinal

4 – Joralemon, Ira B., The Ajo Copper Mining District: Trans. Am. Inst. Min. Eng., vol. 49, 1914, p. 593.

axis of the orebody, strikes about north 30° west and is fairly flat until it plunges downward near its south end (fig. 4). With the exception of the southern portion of the orebody, its shape is such that most of the ore can be mined by open-pit methods with the removal of a relatively small proportion of waste. To strip and mine the ore from the south end of the orebody, an increasing proportion of waste will be handled until the ratio of waste to ore will be such that an underground method of mining will be cheaper.

Strictly speaking, the grade of the ore, as shown by assays of drill hole samples, is irregular; however, the grade of the ore treated is easily kept uniform by mixing ores from various sections of the pit. The average grade of the sulphide ore produced during the year 1930 was 1.24 per cent copper, and the highest and lowest monthly grades during that year were 1.41 and 1.12 per cent copper, respectively.

The ore is on the average very hard, and breaks into large bowlders on account of the presence of widely spaced fracture planes throughout the rock. The central portion and west side of the orebody are highly siliceous, and present a particularly hard drilling and blasting problem. A few minor faults occur within the orebody, but none have been encountered which have seriously affected mining operations. In general, the waste and overburden material is softer than the ore.

METHODS OF PROSPECTING AND EXPLORATION

Drilling

The orebody has been developed almost exclusively by means of diamond drilling. Out of a total of 250 holes drilled on the property, 236 are diamond-drill holes with a total footage of 121,900 feet. The remainder, or 14 holes, were drilled with churn drills, and have a total footage of 4,849 feet. Prior to 1926 most of the holes were drilled to relatively shallow depths; the average depth of 178 diamond-drill holes and the 14 churn-drill holes was only 365 feet. Many of these holes were stopped in ore, or at least in material which was later recognized as ore in view of the improved costs and metallurgical results obtained in the mining and concentration of the sulphide ore; in order to develop the probable extension of the orebody both in depth and laterally, a drilling campaign was started in April, 1926. During this campaign, 58 holes, with an average depth of 976 feet, were drilled.

The majority of the holes in the orebody were drilled vertically downward at the intersections of north-south and east-west coordinates at 200-foot intervals, but in the recent drilling of the deeper ore in the main orebody, 400 feet was adopted as the standard interval between holes. All of the holes were drilled with an A bit, with the exception of a few holes put down with an E bit. The average core recovery for all diamond drilling to date (March, 1931) is 49.8 per cent, with a maximum recovery for any individual hole of 78.4 per cent, and a minimum of 8.3 per cent. Core recoveries above the average were obtained in holes drilled in the western and southern parts of the orebody, while drill holes in the eastern portion of the main orebody gave core recoveries below the average. A greater percentage of core was recovered from deep holes than from shallow holes. Of the total footage of diamond drilling, 62.4 per cent was in monzonite, 21.9 per cent in rhyolite, 7.5 per cent in diorite, 7.2 per cent in agglomerate, and 1 per cent in other formations, mostly small dikes cutting the orebody.

Test Pitting

During the period of the early development of the orebody by diamond drilling, test pits were sunk on drill holes in order to check the results of the drill hole samples; later pits were sunk on drill hole locations in advance of drilling, in order to expedite the development program. Seventy-seven test pits were sunk with a total footage of 3,955 feet, of which 1,234 feet was for checking diamond drilling, and 2,721 feet was in advance of drilling. During the sinking of these test pits, every tenth bucket of broken material windlassed from the hole was taken for a sample. After the test pits were finished, they were sampled on all four sides by means of vertical channels cut 6 inches wide by 3 inches deep. The channel samples averaged 0.15 per cent lower in copper than the bucket samples, and slightly lower than the diamond-drill hole samples.

Drifting and Raising

As most of the test pitting was done in carbonate ore, drifts were run on coordinate lines from the bottom of two of the deepest test pits to check more thoroughly the results of the drilling in sulphide ore. Raises were also put up from the drifts at points where they cut drill holes. The total amount of drifting was 1,513 feet, and of raising, 142 feet.

SAMPLING AND ESTIMATION OF TONNAGES AND VALUES

Drill-Hole Sampling and Calculation of Sample Values

All diamond drilling to date has been done under contract, but the sampling has been done by representatives of the company. The sampling crew consists of a head sampler, and one sampler for each drill shift; the work is under the supervision of the chief engineer. The head sampler directs the work of the samplers at the drills, splits the cores, and makes daily reports to the chief engineer. The samplers are constantly on duty at the drills while drilling is in progress, and attend to the collection of the sludge, drying and sacking of the sludge samples, and measuring and sacking of the cores. With the exception of drilling done in barren capping, samples are taken at intervals of 5 feet. All of the sludge from a run is caught in cylindrical galvanized iron tanks, which are 30 inches in diameter and 26 inches high. Enough tanks are kept at each drill to hold all of the sludge from one run and to give at least one extra tank for starting the next run. Usually five tanks give ample capacity, and when drilling in material which settles quickly four tanks are sufficient. The sludge is allowed to settle as long as practicable, clear water is then siphoned from the tanks, and the thickened sludge is transferred to tubs for drying. Drying is done over a wood fire, but care is taken not to heat the sludge enough to roast the sulphide minerals. After the sludge is dried it is sacked and sent to the assaying laboratory, where it is mixed and samples are cut out for assay. A small sample is returned to the engineering department for filing.

The core which is recovered from a run is measured at the drill and sacked. It is then taken to the sample house, where it is weighed to check the measured lengths, and split lengthwise to give two equal portions of core, one of which is sent to the laboratory for assay and the other retained for geologic study or for a check sample if necessary. A small specimen from the reject half of the core is marked with the hole number and depth and filed for ready geologic reference. These specimens are from 1-1/2 to 2 inches long, and 20 of them, representing 100 feet of drilling, are filed together in one box.

The head sampler makes out a report for each hole, showing the progress for the day, the length of core, rock formation, ore minerals, and hardness of rock for each 5-foot run. These reports are used in conjunction with the assays of the core and sludge reported by the assaying laboratory, to make out the complete logs of the holes. The core and sludge assays are combined according to the relative weight of the core recovered and the calculated weight of sludge on the basis of total sludge recovery. A table showing factors for core lengths from 1 to 60 inches is used to facilitate calculation of the combined assay. Composite samples from every 50 feet of hole are analyzed for iron, alumina, silica, calcium oxide, and sulphur, and assayed for gold and silver. The analyses of the composites are also entered on the drill-hole log.

Test-Pit Sampling

The method of sampling test pits by channels has been discussed under "Prospecting and Exploration."

"Air-Drill" Sampling

Small samples composed of blast-hole drill cuttings are used in part to control production. The cuttings from a blasting hole which is being sampled are caught on a discarded filter blanket placed around the collar of the hole. A galvanized-iron cone about 2 feet in diameter with a hole in the top for the passage of drill steel stops the cuttings when they are blown from the hole and retains them on the blanket. This method of sampling is inexpensive, and can be used without any hindrance or delay to mining operations. This sampling is further discussed under "Estimation."

Estimation of Tonnages and Grades

Horizontal sections at 25-foot intervals, corresponding to present or future mining levels, are used for estimating tonnages and grades of open pit ore. All holes are plotted on the sections which they pierce, and the assays between levels are averaged for all holes and these average assays marked on the section under the hole number. The figures obtained by combining the core and sludge assays are used without further correction. Ore limits and pit limits are drawn on the section and the areas governed by the grades of the various holes are outlined. Each area is then measured with a planimeter which is set so that tonnages are read direct; the tonnage is multiplied by its grade for grade units; the i dividual tonnages and grade units are summed for the level tonnage and grade; and finally, the level tonnages and grade units are summed for total open-pit tonnage and grade. A factor of 12.5 cubic feet per ton is used for conversion of volumes of rock in place to tonnages.

The foregoing method of estimation is not applicable to the southern and southeastern portion of the orebody, which dips steeply toward the south and southeast. This ore is divided into vertical triangular prisms, the tonnage and grade of each prism is calculated, and the sum of the triangular prisms gives the total tonnage.

Diamond drill hole assays are used in estimates for planning mine operations entailing relatively large tonnages, but are not used extensively in the control of the daily produc-ion of ore. Plans for the production of ore for a year, or for a longer period are based entirely on estimates from diamond drill hole assays.

The daily ore production is controlled by the assay of samples, called "air-drill samples," taken from the cuttings of the blasting holes in the benches which are being mined. Monthly plans are made from estimates, using the air-drill assays in conjunction with diamond drill hole assays.

From one to four blasting holes, the number depending on the width of the blast, are sampled at intervals of 20 feet along each bank being drilled. The holes that are sampled are located with a transit and the corresponding assays plotted on the monthly pit map from which daily and monthly ore estimates can be made. The locations of the air-drill holes sampled are marked in the pit with stakes set on the loading tracks opposite the proper lines of holes. Corresponding assays of these samples are on file at the mine office; the loading of ore and waste is controlled by these assays. In addition a daily estimate is made by the engineering department of the ore production from each shovel by using the air-drill samples and the number of cars loaded by the shovel. The mine department each day furnishes the engineering department with the locations of shovels with reference to the sample stakes and the number of cars of ore and waste which were loaded during the previous day's work. The grade of ore obtained by this method of sampling and estimation checks very closely with the grade obtained from the automatic sampling at the concentrator. In the last six months the actual grade of ore, according to automatic samples taken at the concentrator, was slightly less than 1/10 of 1 per cent of copper higher than the grade obtained by air drill hole samples.

CHOICE OF MINING METHOD

Open-pit mining with power shovels is particularly well adapted to the major portion of the New Cornelia orebody. On account of the low grade of the ore, it was necessary to use a method of mining by which the ore could be mined very cheaply. The hardness of the ore alone was enough to decide against caving methods, and the lack of an appreciable amount of overburden made open-pit mining especially attractive. The orebody, being mostly of primary origin, is not as uniform in grade as some of the low-grade porphyry deposits of secondary origin, and consequently a method of mining which permits the separation of ore and waste is particularly desirable. Material below the minimum ore grade can easily be separated in open-pit mining. Plans have been made to mine at least 80 per cent of the developed orebody by open-pit methods, but eventually underground methods will be used to mine a portion of the orebody which has excessive overburden.

In determining the limiting plane between open-pit ore and underground ore, the relative cost of mining by the two methods is the first consideration. For instance, if the cost of mining a ton of ore by underground methods is four times the cost of mining a ton of material (ore or waste) by open-pit methods, a stripping ratio of 3 of waste to 1 of ore could be used in determining the cut-off between underground and open-pit ore. The cost of steam-shovel mining in the New Cornelia pit has been determined by many years of operation, but, as no large-scale underground mining has as yet been attempted, this cost can only be approximated by estimates. A stripping ratio of 3 to 1 has been used in determining the ultimate limits of a pit in places where it was necessary to differentiate between underground and open-pit ore, for the purpose of estimating ore reserves. The pit has at no place approached the stage where in actual mining operations it was necessary to decide between the two methods of mining. Other factors than the relative costs of open-pit and underground mining will also have a bearing on the problem. Lack of dilution of the ore, or separation of waste from ore, flexibility of operation, a greater recovery of ore, and familiarity through years of experience with open-pit methods will be factors in deciding how far power-shovel mining can be carried. A more pertinent problem than that of differentiating between underground ore and open-pit ore is that of determining the limits of the pit in material that can be mined by open-pit methods only, or, in other words, determining the minimum grade of ore that can be mined at a profit. In this case the controlling factors are the total cost of mining and treating a ton of ore of a certain grade, and the value of the copper which can be recovered

from the ore. The total cost of mining and treating a ton of ore will vary according to the grade of the ore for the reason that smelting, refining, and other expenses are dependent upon the metal content of the ore, whereas only mining, transportation of ore, crushing and conveying, and milling expenses are constant for all grades of ore. With the unit costs for these different items, the total costs for various grades of ore can readily be calculated and compared with the amounts that would be received from the sale of the recovered copper at various prices for that metal. The minimum grade of the ore that is to be included in the pit should show a fair operating profit, but material that is mixed with ore above the minimum grade and must be mined in the process of mining the higher grade ore may be treated even though it shows no real profit. In this case, however, milling capacity is the important consideration, for ore that can be treated at a bare profit should not exclude a like amount of ore that would give a fair profit.

MINING METHOD[5]

The original diamond drilling of the orebody which was started in December, 1911, and finished in May, 1913, developed 12,000,000 tons of carbonate ore averaging 1.54 per cent of copper, and 28,000,000 tons of sulphide ore averaging 1.50 per cent of copper. The removal of the carbonate capping from the sulphide ore was the first consideration in the mining of the orebody. Fortunately, it was found that the carbonate ore could be treated profitably by leaching with sulphuric acid solution, followed by precipitation of the copper by electrolysis, so that what might have been purely a stripping problem was turned into a profitable operation. Little or no waste overburden occurred over the carbonate ore. The mining of three carbonate hills, rising with steep slopes to elevations from 115 to 165 feet above the general surface, was the most difficult part of the operation. It was finally decided to attack the hills from their bases and mine them in one lift rather than to try to establish benches at intermediate levels between their bases and crests. The maximum height of bank thus encountered was 160 feet.

Both coyote and churn drill hole blasting were used in breaking the ore in the hills. It was found that coyote blasting was more economical for high banks, over 45 feet, whereas churn drilling was better adapted to low banks. Piston air drills eventually superseded churn drills for drilling banks under 30 feet in height, and these in turn have been replaced by hammer machines. The largest single blast, in which one tunnel and 10 drill holes were shot, broke 285,000 tons of ore.

The broken ore was loaded by means of steam shovels into 20-yard, automatic, air-dump cars and hauled over standard-gage tracks with steam locomotives to the crushing plant. Five or six cars normally constituted a train and the average length of haul was 1 mile.

After the hills had been cut down, mining levels at vertical intervals of approximately 30 feet were established for the purpose of mining the balance of the carbonate ore, but as the contact between the carbonate and sulphide ore was a warped surface, with as much as 50 feet difference in elevation between the high and low points, true levels could not be maintained over the total area of the pit.

Mining of carbonate ore, as stated before, was started in May, 1917, and after about five years of carbonate production, stripping of the sulphide ore was sufficiently advanced to warrant the erection of a concentrating plant. A 5,000-ton concentrator was completed near the close of 1923 and regular production of sulphide ore began in January, 1924. After that time both carbonate and sulphide ore were produced until July, 1930, when the leaching plant was closed and carbonate-ore production ceased. Up to January 1, 1931, 16,800,000

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5 - References in this paper to the scale of present operations refer to the period in 1931, prior to September 1.

tons of carbonate ore and 15,600,000 tons of sulphide ore had been mined by steam shovels from the pit and treated either in the leaching or concentrating plants.

The pit, including the approach, now covers an area of 162 acres, and the bottom level, which is at an altitude of 1,700 feet above sea level, is 317 feet below the top of the highest hill which was mined and 135 feet below the average elevation of the rim. The shape of the pit is roughly elliptical with the major axis pointing northwest-southeast, and is 3,600 feet long by 2,500 feet wide. Four regular levels for the production of sulphides have been opened up, and two additional levels for stripping operations are being maintained.

Entry is made into the pit at its south end with an approach which starts at a point about 2,000 feet from the crushing plants and parallels the east side of the pit at a distance of about 1,000 feet to the east. This approach carries all main tracks connecting with the ore levels. The track in the bottom of the approach is on a 2 per cent adverse grade. Waste as well as ore is hauled through this approach, but is carried over a separate line from the mouth of the approach to the waste dumps. It is planned to connect the upper levels at the south end of the pit with an independent line to the dumps; it is estimated that a large portion of the stripping tonnage can be thus hauled out of the pit without interference with ore haulage.

Stripping

To date the stripping of waste overburden has not been an important factor in the mining of ore. To January 1, 1931, only 6,800,000 tons of waste had been removed in the mining of 32,400,000 tons of ore, a ratio of 0.21 tons of waste to 1 ton of ore. A large portion of this material occurred within the orebody and in the approach, and strictly speaking was not overburden. Stripping will become an important factor when the south end of the orebody, which dips under a barren capping, is uncovered and mined. A waste ratio of about 0.3 ton of waste per ton of ore is being maintained at the present time, and it is estimated that the total remaining open-pit ore as now known will be mined with a ratio of 0.8 ton of waste to each ton of ore. No special equipment is used in the mining and haulage of waste; the steam shovels, dump cars, and locomotives are the same as used for ore production.

The main waste dumps are located on the east side of a low range of hills which lies a short distance east of the pit, where an area of nearly 1 square mile is available for dumping purposes if needed. The material on these dumps contains 0.30 per cent copper, mostly in the form of chalcopyrite.

Mining

The average daily production during the time when both sulphide and carbonate ores were being treated was 9,500 tons --6,500 tons of sulphide and 3,000 tons of carbonate ore. Since July, 1930, only sulphide ore has been mined, and the average daily rate has been about 6,500 tons. The present production, including waste, is obtained in 7 shovel shifts -- 3 shovels working during the day shift and 4 during the night shift.

The active mining benches have an average length of about 2,000 feet and an average width of 350 feet. The general practice has been to mine the benches on the east and west sides of the pit separately, and no attempt is made to maintain a bench for the entire distance around the pit. This system evolved naturally from the mining of the carbonates and the opening up of the first two levels in sulphides through an approach which entered the north end of the pit and roughly cut the ore area into two parts, one east and the other west of the approach. After the new approach was connected to the south end of the pit which is at the narrow end of the orebody, the old central approach was mined out and benches on one side

of the pit were connected with those of corresponding elevation on the other side. The benches are roughly circular at their north ends (fig. 7).

The height of the existing banks ranges from 25 to 35 feet, but 25 feet has been determined as the most economical height, and is being used in opening up new levels. The average slope of the banks is 50° from the horizontal; however, 25-foot banks in average ground will stand nearly vertical if properly trimmed. Occasionally fault planes paralleling the bank and dipping toward the pit determine the bank slope.

The pit has not progressed to the stage where the maximum slope for the sides of the pit can actually be determined; however, it is assumed, for the purpose of making estimates of open-pit ore, that a slope of 45° can be safely maintained. This will be the slope from the rim of the pit to the toe of the bank at the bottom, and will include berms for the protection of the working levels against minor rock slides.

DRILLING AND BLASTING

Each bank in the pit presents its own blasting problem; however, there are definite objects and fundamental limits that are common to all blasts. The principal object of a primary blast is to break as much ground as possible into material which can easily be handled by the shovels; the size of the blast is limited principally by the number of exploders that can safely be detonated at once and by the length of time required to load the holes.

In general, the larger the area covered by the primary blast, the less will be the proportion of material requiring secondary blasting before it can be shoveled, and consequently the cheaper will be the total average cost of breaking the ground. It is advantageous to break as long a section as possible with a single blast, for the reason that there usually is a rib of poorly broken ground between contiguous sections of a bank which have been shot, and consequently the number of these ribs will depend upon the length of the sections blasted. A wide blast, especially in very hard ground, is also desirable for the reason that it gives much better fragmentation than a narrow one, due to the fact that the powder in the center of the shot is more evenly distributed and shatters the ground better than that at either face or bank side of the blast. The face opens up along seams and is thrown forward, and large bowlders, which require secondary blasting, remain on or near the surface. Likewise, a strip of ground along the bank side of the blast breaks along major fractures, and requires secondary blasting before it can be shoveled. The maximum area covered by a single blast will vary from 90 to 100 feet in width and from 200 to 300 feet in length.

The time required for loading the holes is important, especially in wet ground, as in this case there is an appreciable loss in strength of powder from the time it is loaded until it is detonated.

Cushion blasting -- that is, blasting behind ground already shot -- is seldom used, for the reason that locating holes along a bank that is not faced up is largely guesswork, and unless the rock is quite soft, a poorly broken rib for the entire length of the blast is likely to result from the procedure.

Drilling

It was early demonstrated that air drills were more satisfactory than churn drills for drilling the hard rock encountered in the orebody, especially when the depth of holes did not much exceed 30 feet, and as soon as regular benches could be established, air drills were adopted as standard. The original air drills were piston machines with tripod mountings, and were used successfully until replaced by hammer drills. Drilling tests made in 1923 with hammer drills established the fact that a much greater footage could be obtained with them

than with piston machines, and several years' experience since then with the hammer type of drill have established its superiority over any other type of drill used. Snce 1923 the piston drills were gradually replaced, and at the present time hammer drills are used exclusively. The average rate of drilling with piston machines was about 24 feet per shift, whereas the average rate with hammer drills is about 80 feet per shift. During a 6-month period extending from November, 1930, to April, 1931, 190,264 feet were drilled in 2,306 drill shifts, or an average of 82.5 feet per shift. No mountings are used with these drills, but they are equipped with spring handles, and when in operation they are suspended from tripods with block and tackle for ease of handling. A tripod consists of three 18-foot pieces of 1-1/2-inch pipe flattened at the upper ends and bolted together with a clevis. The drill is suspended from the clevis with a twofold tackle and can readily be hauled up when it is necessary either to free the steel in the hole or to change steel. An air pressure of 100 pounds per square inch is maintained at the compressors, or between 90 and 95 pounds at the drills. Drill steel is made from 1-1/4-inch hollow-round steel in lengths from 4 feet for starters up to 36 feet for the longest drill with a difference of 2 feet for each change. Cross bits are used. The gage of starter bits is 2-7/16 inches and a decrease in gage of 1/16-inch is made for each change in length.

All holes are drilled 2 feet below grade so that when they are shot no hard ribs or knobs will be left above grade. Spacing of the holes varies with the hardness of the rock and the height of the bank. For a 25-foot bank this spacing ranges from 10 feet in the hardest ground to 20 feet in the softest. Figure 6 shows the arrangement of holes for a standard 25-foot bank; for lower banks the distance between holes would be proportionately less. The distance from the first row of holes to the crest of the bank is less than the distance between succeeding rows, in order to take care of the slope of the bank. Should this slope be very flat, short toe holes are drilled to relieve the burden on the vertical holes. The front row is used as a base, and from it the others are located, the holes in alternate rows being staggered (fig. 6).

A drill crew consists of two machinemen. At the present 18 crews are employed, 9 crews working day shifts and 9 crews on night shifts. Besides the machinemen, 8 muckers are employed on day shift to clear away loose rock around the points where holes are to be drilled and to set casing wherever it is required. A drill foreman lays out and supervises the work of the drill crews and muckers.

All banks are carried on a predetermined grade and grade stakes are set by a level party at 50-foot intervals along a bank which is to be drilled. The drill foreman then marks the position and depth of holes to be drilled, and sets the muckers preparing these sites for the drill crews. In case the ground is fractured from previous blasts, short lengths of 2-1/2-inch pipe are set for casing the collar of the hole. When the section of the bank which is to be shot in one blast has been drilled, drilling is continued into the adjacent section, but a strip of undrilled ground is left between the two. This prevents loss of holes when the first section is blasted, and additional holes can be drilled afterwards, if necessary, and shot with the next blast.

Blasting

All holes are sprung with 1-1/8-inch 50 per cent strength gelatin stick powder; 40 per cent strength granular powder, known as Quarry Special No. 1, is used for the main charges in dry ground, and 1-1/8-inch 40 per cent strength gelatin stick explosive in wet ground. Forty per cent strength gelatin stick powder is also used for the main charges in toe holes. Granular powder is fairly free running and has an advantage over stick powder in that it can be loaded much faster.

Primers are made of a single stick of 1-1/8-inch 50 per cent strength gelatin powder in which one No. 8 electric detonator is inserted. The detonator is inserted in one end of the stick, and the lead wires brought up along the sides of the primer to the end opposite the detonator, where a half-hitch is taken with each wire. The hitches are made so that the free ends of the wire will be on opposite sides of the primer and it will hang vertically with the cap at the lower end when lowered into the hole. After the hole is about 80 per cent loaded, the primer is lowered and pressed into the charge, and the remainder of the charge is then placed on top of the primer, thus insuring direct contact of the primer with the main charge. Two primers are used in holes in ravelly ground, one placed at the bottom of the pocket and the other at the top of the charge.

Pockets for the main charge are made at the bottom of the drill holes by springing them with 50 per cent strength gelatin stick powder. In this process the hole is deepened a few inches, as it is best to have the pocket well below grade. The number of springings required to form a pocket of sufficient size to hold the charge varies with the hardness of the ground. Two springings usually are sufficient, but in some ground as many as six are necessary. The ratio of the amount of powder used in any springing charge to that necessary to fill the pocket after it is sprung is fairly constant for a particular hole, so that by determining this ratio for the first springing it can readily be calculated how many times the hole need be sprung in order to hold the final charge. If, for instance, two sticks of powder used in the first springing makes a pocket large enough for 10 sticks of powder, the ratio is 1 to 5, and the second springing will then make a pocket large enough for 50 sticks. If the load for the hole is equivalent to 50 sticks or less, two springings will, therefore, be sufficient; if the load is more, additional springing will be necessary.

The first springing charge, which usually is 2 sticks of powder, is stemmed with tailings sand for a distance of about 2 feet above the charge. After shooting, the hole is blown out by compressed air. Succeeding springing charges are stemmed with greater amounts of sand Springing is carried on in rotation over as large an area as practicable, so that after blowing out the holes, sufficient time has elapsed for the holes first sprung to be cool enough for another charge. All of the holes of the blast are sprung before the final loading of the holes is started.

Granular powder, which is used for the main charge, is poured directly in the holes and lightly tamped with wooden tamping sticks to insure filling the pockets. After the primers have been placed in the charge and the pockets have been filled with powder the holes are stemmed to the top with dirt. In very hard ground, where the holes are more than 20 feet deep, an auxiliary load consisting of from 10 to 20 sticks of 50 per cent gelatin powder is placed about halfway up the hole to insure thorough breaking of the rock near the surface. Wherever this auxiliary load is used, two primers are always placed in the main charge and one primer in the auxiliary charge. When all holes are loaded, lead wires of No. 10 insulated copper wire are strung between the holes and tested. The exploders are then connected in parallel to the lead wires, and the blast fired by current from a 110-volt power line; the firing switch is located near the main line at a safe distance from the blast. After the bank has been shot, it is inspected, and loose casings and wires are removed.

The total amount of powder used in a primary blast varies with the hardness of the rock and the burden on the blast. As a guide for loading, 1/2-pound of powder is used per ton for hard rock, 1/3-pound per ton for medium rock, and 1/4-pound per ton for soft rock. This is a slight overload, but with this as a base and with some experience the right amount can be determined for each class of rock. The calculated loads do not take into consideration any overbreak, and as there is always a large tonnage broken beyond theoretical lines, the actual tonnage broken per pound of powder will be greater than that calculated. Due to improved blasting practice and lower banks, the amount of material broken per pound of powder has

shown a decided increase during the past 10 years of mining as shown in Table 1. No separate account is made of powder used in primary and secondary blasting, and the above figures are on the basis of total powder consumption.

As the spacing of the holes drilled for blasting is determined by the hardness of the rock, the amount of powder used in any hole will depend principally on its depth. About one box or 50 pounds of powder is used as the main charge in a 20-foot hole; 1-3/4 to 2 boxes in a 25-foot hole; and 3-1/4 to 3-3/4 boxes in a 30-foot hole. The charges in the row of holes nearest the face of the bank vary somewhat due to irregularity of the toe, but all back holes of a blast are loaded with the same amounts of powder.

The material broken by a primary blast is comparatively coarse, as stated before, because of the presence of widely spaced fractures in the rock formation; roughly one-half of the broken ore will be in pieces larger than 18-inch cubes. Before the ore can be loaded into cars it must be broken to a size to pass the shovel dipper, which has a clearance of 4 feet by 3 feet. Bailing oversize bowlders into the cars is not permissible for the reason that the primary crushers will handle no larger pieces of ore than will pass through the dipper. One jackhammer man at each shovel blockholes the large bowlders as they are encountered in the shoveling of the ore. A hole is drilled with a light jackhammer to the center of the bowlder, and blasted with from 1/4 to 1/2 stick of 40 per cent gelatin powder.

Table 1.- Tons of all material (ore and waste) broken per pound of powder consumed

Year	Dry tons per pound
1921	3.05
1922	3.27
1923	2.67
1924	2.27
1925	3.37
1926	3.93
1927	4.22
1928	4.39
1929	5.85
1930	5.93

LOADING THE ORE AND WASTE

Ore and waste are loaded by means of 105-ton railroad-type steam shovels fueled with oil and equipped with 4-yard dippers, 20-foot 6-inch dipper sticks, and 30-foot booms. Ten of these shovels are now available for operation. This type has been used since the beginning of shovel operations, and although many minor changes in the design of the shovels have been made from time to time to adapt them to the particular conditions at Ajo, the general specifications of all shovels are the same. The fact that the first shovel placed in operation is as serviceable now as it was originally, after 14 years of almost continuous use, is evidence of their ruggedness as well as the care with which they have been serviced and repaired.

Waste is shoveled with the same equipment as is used for ore, and the only difference in the two operations is that the waste trains are routed to the waste dumps instead of to the crushers.

From a study of the air-drill assays, the mine superintendent designates any portion of banks which are to be wasted. The method of ore control is more fully described under "Sampling." By this method a close separation of ore and waste is easily made, especially when shoveling in a bank not exceeding 25 feet in height.

The shovel crew consists of one runner, one craneman, one fireman, and four pitmen. One of the pitmen is also a jackhammer man and does all blockholing of bowlders in front of the shovel and hard nobs in the bottom of the cut.

The maximum width from which a shovel can load is 48 feet, measured from the inside rail of the loading track to the toe of the cut. This is equivalent to a cut of approximately 30 feet in the solid bank. The maximum height at which the dipper can dump its load is 18-1/2 feet above the section rails, which limits the depth of drop cuts to about 10 feet.

The shovels move on 6-foot sections of track made of 70-pound rails; the shovel track is laid parallel and at a distance of 23 feet from the loading track. When the end of a cut is reached, the shovel is moved back on the loading track to a new position, and the loading track is either lined over to the blasted bank, or is torn up and relaid. The latter method usually is employed, since it saves considerable grading and gives a better track. Some track is laid in panels with a locomotive crane, but this method is never as satisfactory as relaid tracks with staggered joints.

A new level is established by making successive drop cuts from the old level until the required elevation has been reached. The first drop is carried on a 2 per cent down grade until a depth of 10 feet has been reached, from which point it is carried forward 10 feet below the original level. When this cut has been finished, the loading track is laid in its bottom and the second cut is then made in either bank. The second cut is made in the same manner as the first, excepting that the grade is continued downward until a depth of 10 feet below the new loading track elevation has been reached. Thus in establishing a level 25 feet below the old level, three drop cuts are necessary.

Fueling of shovels is accomplished by attaching a 3-inch hose to the oil tank on the locomotive tender by special flange connection and pumping oil from the locomotive to the fuel-oil tank on the shovel. The shovel tank has a capacity for 1-1/2 shifts, and it usually takes from 5 to 10 minutes to refuel. The tank on each locomotive has a capacity sufficient to refuel one shovel and in addition to run the locomotive for an 8-hour shift.

Minor repairs to shovels are made by the operating crews between trains; heavy repairs and moving of shovels are handled by the repair crew. Shovels usually are washed out once a month by the repair crew and are completely overhauled about once every five years.

During the year 1930 a total of 3,410,114 dry tons of material was loaded in 2,580 shovel-shifts, or an average rate of 1,322 dry tons per shovel-shift. Since the beginning of operations the maximum amount of material handled by any shovel during a shift was 107 cars, or approximately 4,200 tons.

TRANSPORTATION

The transportation equipment for hauling ore and waste from the pit consists of 15 oil-fired steam locomotives and 82, 20-yard, automatic, air-dump cars. The locomotives are class 060 S134. The weight of a locomotive alone is 134,000 pounds, and that of the tender 88,800 pounds. The maximum tractive power is 28,700 pounds. The tender is 8-wheeled and has a capacity of 4,000 gallons of water and 1,500 gallons of oil. The locomotive boiler is designed for 180 pounds steam pressure.

The dump cars are flat-bottomed and can be dumped from either side. The over-all length of a car is 32 feet, and outside width of the body is 10 feet 2 inches. The height from the top of the rail to the top of the car is 8 feet 5-1/2 inches. Five or six cars normally constitute a train, the average load for a car being 40 tons of ore or waste.

The present track system from pit to crushers and dumps has a total length of 13 miles of standard-gage track, 10 miles of which is in the pit and approach to the pit, 2 miles on the waste dump and approach, and 1 mile on the main line from the pit approach to the crushing plants. The maximum adverse grade on tracks is 2 per cent, compensated on curves, and the maximum curvature on main lines is 20°. All tracks are laid with 70-pound rails and 8 by 8 inch by 8-foot cross ties. Most of the shifting of tracks is done with a track shifter. The track gang consists of 25 men, supervised by a foreman and two subforemen.

Trains are hauled from the shovels, where they are loaded, directly to the crushing plants or dumps. The average length of haul to the crushing plants is 1.5 miles, and to the waste dumps 1.8 miles.

DRAINAGE

A system of dams and ditches diverts rainfall run-off away from the pit, so that only water that actually falls within the pit area need be handled. The lowest level in the pit has a slight slope to a central low point, where an air-operated pump with a capacity of 75 gallons per minute lifts the water over the edge of the pit to the outside drainage system. As the average rainfall at Ajo is under 10 inches annually, very little inconvenience is caused by water collecting in the pit, excepting at infrequent times when very heavy rains occur.

LIGHTING

The shovels are equipped with electric lights for night work. The current for the shovel lights, as well as lights for the drill crews, is obtained from the main power line into the pit, which also supplies current for blasting.

PLANTS

A 15,000-kilowatt steam turbine electric power plant, near the crushing plants and machine shop, furnishes all electric power for the mine, plant, and townsite. The mine uses less than 5 per cent of the total power generated. The compressor and drill-sharpening shop uses 60 per cent of the electric power consumed at the mine, and the remainder is consumed mainly by lighting. A 2,300-volt line connects the compressor plant with the power house.

Water for use at the mine, plant, and townsite is pumped from a well 7 miles north of Ajo to storage tanks above the concentrator. A 6-inch pipe line carries water from the storage tanks to the mine.

The compressor plant, drill-sharpening shop, and shop for minor repairs are at the mouth of the approach to the pit in close proximity to the main tracks.

The compressor equipment consists of three angle compound compressors having a total capacity of 6,200 cubic feet of free air per minute. Each compressor is composed of two independent 2-stage units direct-connected with a synchronous motor. A pressure of 100 pounds per square inch is maintained in the air receivers at the compressor plant.

The drill-sharpening shop is equipped with two sharpeners, one for sharpening the primary drilling steel, and the other for the secondary or plugger steel, and three oil furnaces, two for heating steel and one for tempering. Sharpened steel is delivered to the working places and dull steel returned to the shop in a specially designed flat car which is hauled by a gasoline-operated track shifter. The track shifter is in use for about 8 hours each day for delivering steel and powder; the balance of the time it is available for track shifting.

The powder magazine is located near the waste dump approach track at a safe distance from the plant and mine. Eighteen-inch adobe walls and a 4-inch layer of sand over the ceiling makes the powder house practically bulletproof.

PER CENT OF EXTRACTION

The total tonnage of ore mined and treated to date is over 5 per cent in excess of that estimated from diamond-drilling data with no appreciable difference in the average grades of ore as estimated and treated.

WAGE, CONTRACT, AND BONUS SYSTEMS

All labor is paid according to a daily wage system, and no bonus or contract system is in effect. The following tabulation shows the average wage scale for 1930 for labor at the mine:

Shovel runners	$8.15
Cranemen	5.70
Powdermen	4.66
Machine drill runners	4.48
Firemen	4.37
Jackhammer runners and powdermen helpers	3.14
Pitmen	2.74
Trackmen and muckers	2.56

Table 2.- Summary of costs

		Year 1930
Ore loaded during period	dry tons	2,376,764
Waste loaded during period	do.	1,040,610
All material loaded during period	do.	3,417,374

Operating costs per dry ton of all material mined

	(1)	(2)	(3)	(4)	(5)	(6)	(7)	(8)
Drilling and blasting	$0.035	$0.003	$0.001	$0.003	$0.001	$0.025	$0.002	$0.070
Loading	.026	.001	-	.001	.013	-	.008	.049
Transportation	.042	.001	-	-	.016	-	.007	.066
Miscellaneous	.001	.016	-	-	-	-	008	025
Total	0.104	0.021	0.001	0.004	0.030	0.025	0.025	0.210

(1) = Labor
(2) = Supervision
(3) = Compressed air, drills, and steel
(4) = Electric power
(5) = Fuel
(6) = Explosives
(7) = Other supplies
(8) = Total

GENERAL COST DATA

Labor:		
	Total man-shifts for year	80,994
	Dry tons, all material mined per man-shift	42.19
	Man-hours per dry ton, all material	0.19
Explosives:		
	Pounds of powder consumed during year	576,150
	Dry tons, all material mined per pound of powder	5.93
Power:		
	Total kw.h. used during the year	1,903,947
	Dry tons all material mined per kw. h.	1.79
Fuel:		
	Barrels of fuel oil consumed by steam shovels during year	26,956
	Barrels of fuel oil consumed by locomotives during year	28,387
	Total barrels fuel oil consumed	55,343
	Dry tons all material mined per barrel fuel oil	61.75

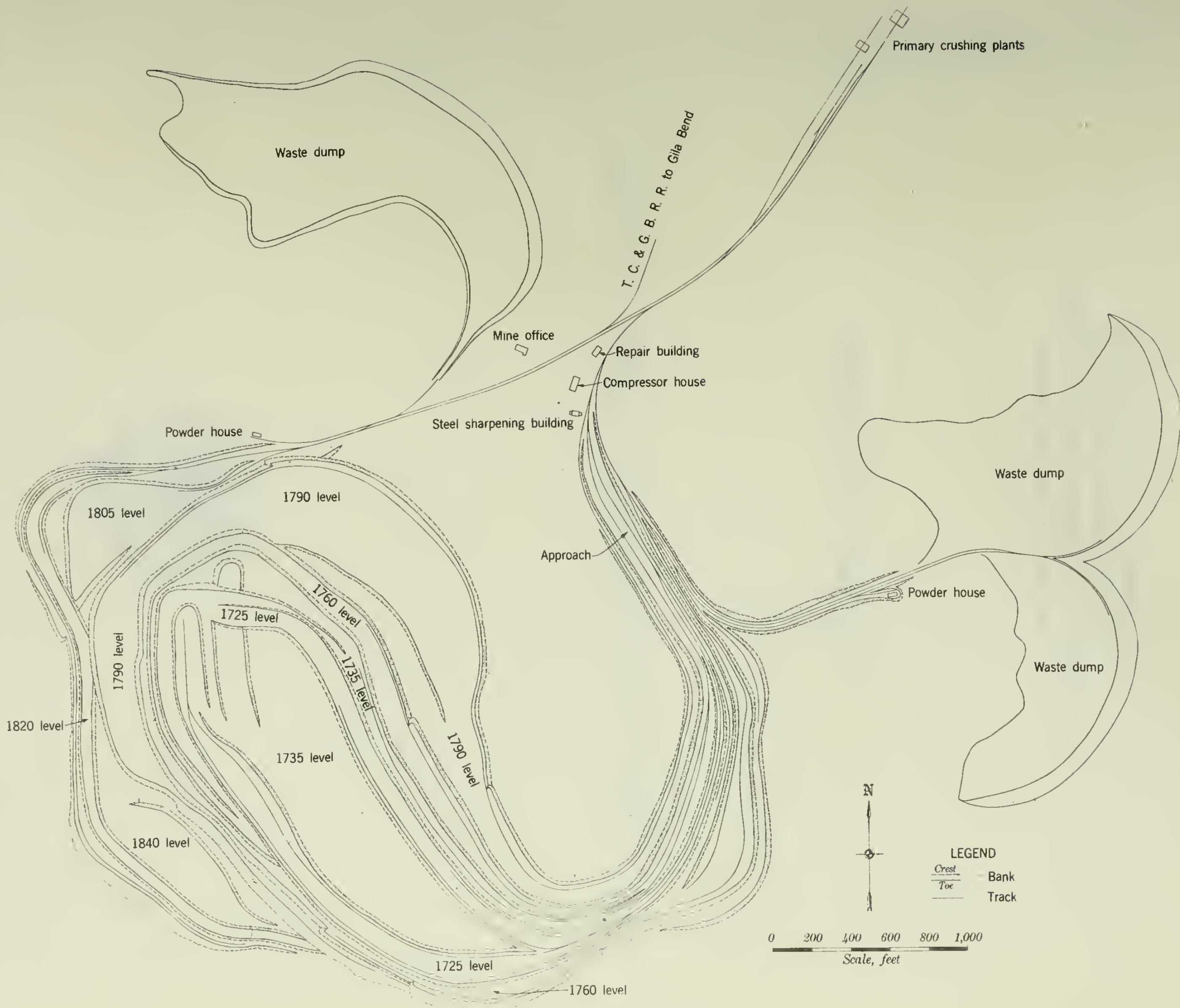

Figure 7.—Plan of New Cornelia pit

I. C. 6667

October, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES

Scott Turner, Director

INFORMATION CIRCULAR

RADIUM IN MEDICAL USE IN THE UNITED STATES

BY

R. R. Sayers

I.C.6667,
October, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

RADIUM IN MEDICAL USE IN THE UNITED STATES[1]

By R. R. Sayers[2]

Radioactivity, the property of radium that led to its isolation more than 30 years ago, is the characteristic that makes it of value in the treatment of disease. Tyler[3] gives the following brief history of the investigation:

> The way to the discovery of radium was opened in 1895 by Röntgen, who found that the glow from a Crooke's tube contained penetrating rays, which he called X rays. Prof. Henri Bequerel, while investigating the effect of various phosphorescent substances, found that uranium salts produced photographic impressions even when enveloped with opaque substances. To Marie Sklovouski, a young Polish student, who later became Madame Curie, Professor Bacquerel delegated the task of learning how and why uranium possessed power to emit these peculiar rays, which he had proved to be electrical in character. Madam Curie, examining by electrical methods the radioactivity of a large number of minerals containing uranium and thorium, discovered that some specimens of pitchblende had about four times the activity of the metal uranium; that chalcolite, the crystallized phosphate of copper and uranium, was twice as active as uranium; that autunite, a phosphate of calcium and uranium, was quite as active as the same weight of pure uranium. In order to check these discoveries, she prepared chalcolite artificially, starting with pure products, but found that this artificial chalcolite had only the activity represented by its composition, or, roughly, 40 per cent of the activity of uranium. This led to the conclusion that there was some element or substance in the residue from uranium minerals that possesses a high degree of activity. After an exhaustive chemical investigation of pitchblende from Joachimsthal, she found that this mineral contained not only uranium but also another radioactive substance, to which she gave the name of polonium, in honor of her native land. Later in 1898, Monsieur and Madame Curie found still another element, which, when brought to a state of concentration, was several million times as active as uranium, and to this was given the name of radium. Debierne afterwards found a fifth radioactive substance, actinium; and in 1906 Boltwood isolated the metal ionium. Strictly pure radium chloride was first produced in 1902.

The first radium was produced commercially from the uranium residues obtained from the mines of Joachimsthal, Bohemia. Tyler calls to attention that, as the ores were a government monopoly, search was begun at once for sources in other parts of the world. As a result, radium-containing ores have been found in about 10 countries.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6667."

Presented before the meeting of American Society of Radiologists, St. Louis, December 1, 1931.

2 - Chief surgeon, U. S. Bureau of Mines; surgeon, U. S. Public Health Service.

3 - Tyler, Paul M., Radium: Inf. Cir. 6312, Bureau of Mines, 1930, 55 pp.

The interest of the Bureau of Mines in the efficient recovery of radium extends back to about 1912. Under an agreement with the National Radium Institute[4] the bureau built and, in June, 1914, began the operation of a radium recovery plant at Denver, Colo. By the time the work ceased in January, 1917, 8.5 grams of radium had been produced. The methods devised by this investigation reduced the cost of recovering radium to about one-third of the then current prices. F. L. Hess, principal mineral technologist of the Bureau of Mines, states that at present only two countries are producing significant quantities of radium - the Belgian Congo at Chinkolobwe, Katanga, and Czechoslovakia at Jachymov (formerly St. Joachimsthal). In 1922 there was an important development in the radium industry by the opening in that year of the radium works of the Société Metallurgique de Hoboken at Oolen, in Belgium, which soon acquired a monopolistic position. The material treated in these works was obtained from the Belgian Congo through the Union Minière du Haut Katanga. In the concessions of this company, which is primarily copper-producing, radium-bearing ore was discovered at Luiswishi in 1913 and at Chinkolobwe in 1915. ********** The present productive capacity of the Oolen works is about 6 grams of radium monthly. At present, the price fluctuates between 60 and 70 dollars per milligram (pounds 12 10s. to pounds 15), or is in the region of 270 marks.[6] These two countries now produce annually about 60 grams and 3.5 grams, respectively. Besides these, seven other countries have radium deposits which, in order of size of the known deposits, are about as follows: United States, Canada, Russia, Portugal, Madagascar, England, and Australia. It is possible that the position of the first-named countries may be reversed before another year has passed. The deposits discovered in 1930 on the east side of Great Bear Lake, District of Mackenzie, Canada, are said to be the richest and possibly the largest yet found.[7] From prospecting operations during 1931, 20 tons of pitchblende were obtained and shipped The extractions that have been made from batches of this ore have been well over 90 per cent. The 20 tons is said to contain at least $100,000 worth of radium at present prices.

During the past few years, the amount of radium now used for medical purposes has been much discussed. In Great Britain plans for distributing and supervising the supply of radium for treatment of disease have been completed and put into effect. The first annual report[8] states that the National Radium Trust has as its chief duties: To take charge of funds raised by public subscription and voted by Parliament for the provision of radium and to arrange for the purchase of radium. The Radium Commission has the duty of making the arrangement for the proper custody, equitable distribution, and full use of the radium purchased by the Radium Trust. The Radium Commission is composed of 10 members and a chairman. The trust appoints the chairman. The Minister of Health, the Secretary of State for Scotland, the Medical Research Council, and the Department of Scientific and Industrial Research severally appoint one of the members. The remaining six members are selected by the trust from a panel of not less than 12 persons having special knowledge and experience in the application of radium in the treatment of the sick. Up to August, 1930, the National Radium Trust had ordered 18.5 grams of radium. Of this amount some 18 grams had been provisionally allocated to national centers; only 14 grams had been received from the radium manufacturers, and not more than 8 grams had been actually delivered to the centers. In France[9] the government has

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4 - The National Radium Institute (Inc.), was a corporation organized and existing under and by virtue of the laws of the State of Delaware for the study of the best methods of producing uranium, vanadium, and radium.

5 - This statement is contained in a chapter prepared for publication in Spurr and Wormser's Marketing of Metals and Minerals.

6 - Canadian Mining Journal, The Belgian Radium Industry: June, 1932, pp. 255-256.

7 - Goodwin, W. M., Mines Branch Radium Plant: Canadian Min. Jour., June, 1932, pp. 253-255.

8 - National Radium Trust and Radium Commission, First Annual Reports, 1929-1930: London, 1930, 74 pp.

9 - See reference 3.

established 15 radium centers in various parts of the country, and allocated to them 31.5 grams of radium (in addition to about 20 grams believed to be privately owned). In Sweden[10] the use of radium is highly organized under Government auspices. A strong movement was started in 1929 to acquire a larger supply of radium in England, where a governmental committee recommended that 24 grams be acquired in addition to about 25 grams then available in the British Isles. In Germany[11] –

> The federal commission for the control of cancer summoned experts recently to discuss the question of the purchase and distribution of radioactive substances such as radium and mesothorium. As a result, the Deutsches Zentralkomittee zur Erforschung und Bekämpfung der Krebskrankheit and the Reichausschuss für Krebsbekämpfung agreed on certain criteria, which are set forth in a communication by Professor Friedrich: The purchase of radioactive substances should be considered only by such institutions as have an experienced radiologist for the therapeutic application of such substances. The use of small quantities of radioactive substance for the local treatment of carcinomas is the method indicated bythe present status of medical knowledge, whereas the treatment a distance with heavy doses of several grams is a method concerning which no definitive opinion can be given as yet. Supplies of radioactive substances should be available in all university clinics and large hospitals, the amount ranging from 200 to 500 milligrams, according to the population of the area for which provision is made. Some centers should have an additional amount in order that they may serve also as research centers. Such central institutes should have an average of 50 milligrams of radioactive substance for each bed designed solely for patients to be irradiated. A central institute should have not less than 20 beds set apart for the use of cancer patients to be irradiated with radioactive substances, which presupposes accordingly the possession of 1,000 milligrams of radioactive substance. Since experience has shown that the biologic effects of the rays emanating from radium and mesothorium are of equal value, in the purchase of supplies the choice as between radium or mesothorium should depend exclusively on economic and practical considerations. The legal regulation of the application of radioactive substances for therapeutic purposes is a problem of the near future.

In the United States, from the time that American radium factories made the first production of radium salts in 1913 to the last recorded output in 1926, they isolated about 203.3 grams of radium. Besides this production the following imports of radium in salts are shown by the customs records. It is thought that at least 3 grams were imported previous to 1923, at which time Belgian radium began to arrive in this country. Production and imports are shown in the following table:

10 – See reference 3.

11 – Journal of the American Medical Association, The Purchase and Distribution of Radioactive Substances: Foreign Letters, Berlin, vol. 97, No. 14, October 3, 1931, p. 1016.

Year	Radium produced in the United States, grams	Radium imported into the United States, grams
1913	2.1	-
1914	9.6	-
1915	4.71	-
1916	8.17	-
1917	13.83	3.00 (estimated)
1918	22.79	-
1919	28.648	-
1920	32.539	-
1921	35.693	-
1922	24.189	-
1923	12.212	8.75
1924	3.365	8.1
1925	2.952	9.27
1926	1.725	10.97
1927	none	7.26
1928	none	10.97
1929	none	10.69
1930	none	16.86
Total	202.523	85.87

The total production and imports into this country to the end of 1930 have been in the neighborhood of 288.4 grams. It is probable that since 1916, including use during the Great War, not more than an average of 2 grams per year has been used in luminous materials, a total of not more than 30 grams. What exports have amounted to is unknown, but they have probably not exceeded 20 grams, so that, making no allowance for broken tubes and other losses there would appear to be still in this country 238 grams of radium. This is very much more than can be accounted for from holdings, and there may have been much larger exports than have been recorded.

The amount of radium in the United States now used for medical purposes has been variously estimated at 50 to well over 200 grams. The following statement was published in the Journal of the American Medical Association:[12]

> According to figures supplied by the American Society for the Conrol of Cancer, which are recognized to be only relatively complete, the total amount of radium owned in quantities of 75 milligrams and over in the United States is 85,-228 milligrams. The hospitals owning 75 milligrams and over number 135. The 135 hospitals own 68,033 milligrams. The individuals owning 75 milligrams and over total 47, with a total of 6,945 milligrams.

The United States Bureau of Mines recently sent out to all the hospitals listed in the 1931 Directory of the American Medical Association the following letter and questionnaire, A and B.

12 - Journal of the American Medical Association, Radium Owned by Hospitals and Physicians: Vol. 96, No. 24, June 13, 1931, p. 2057.

UNITED STATES
DEPARTMENT OF COMMERCE
BUREAU OF MINES
WASHINGTON

The Bureau of Mines is asked repeatedly as to the need for additional radium in the United States. The question naturally arises as to how much is now available, the number of people receiving treatment, and the total number of treatments needed.

In order to more accurately supply this information, the inclosed questionnaire is being sent to the principal hospitals in the United States. I would very much appreciate it if you would have this form filled out as completely as possible and returned at once in the inclosed franked envelope.

Yours sincerely,

R. R. SAYERS,
Chief Surgeon, U.S.B. of M.
Surgeon, U.S.P.H.S.

Inclosures

DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR
WASHINGTON

CONFIDENTIAL
FOR GOVERNMENT
USE ONLY

RADIUM - FOR MEDICAL USE

1. How much radium (element) have you on hand .... Grams .... Milligrams

2. Please show below quantity of radium in each salt:

Salt	Grams	Milligrams	Salt	Grams	Milligrams
Bromide......	....	....	Carbonate......	....	....
Sulphate....	....	....	Other(Please		
Chloride....	....	....	specify)......	....	....

3. Date when acquired ....

4. Please show below quantity of radium in each form:

Form	Grams	Milligrams
Applicators....................................	....	....
Needles....................................	....	....
Tubes....................................	....	....
Solution....................................	....	....

5. Number of patients treated annually ....

6. Total number of treatments ....

7. If you do not have radium, how is radium treatment obtained for patients in your hospital or in your vicinity who need such treatment? ....
....

8. What is the usual price paid per milligram hour by your hospital or by individuals to institutions or persons furnishing radium treatment? ....
....

9. Are these arrangements satisfactory to your institution? ....

10. How much more radium could you use to advantage? ....
(This question is intended only to give the United States Bureau of Mines an idea of the country's need for more radium)

11. Do you know of physicians outside of hospital having radium in your city? If so, please give names on reverse of this sheet.

....
(Please sign here)
(Official title).....................

(B)

Figure 1 — Distribution of radium for medical use in the United States, milligrams

To date 287 hospitals and clinics have reported that they have a total of 85,800.26 milligrams of radium, and 128 of the 287 each had 75 milligrams or more; 414 physicians have reported that they have 33,286.93 milligrams and 171 of the 414 each had 75 milligrams or more; 9 laboratories and companies have reported that they have 5,545.42 milligrams and 5 of the 9 each had 75 milligrams or more. New York, as would be expected, had the largest amount - 29,800.61 milligrams; Pennsylvania had the next largest amount - 12,902.21 milligrams. Five States reported no radium in hospitals, and according to the reports received, no radium is owned in one State. (See fig. 1.)

All of the reports did not designate the kind of salt in which the radium was held. Those reporting, however, gave the following quantities:

	Milligrams
Bromide	24,676.44
Sulphate	49,939.23
Chloride	17,047.11
Carbonate	169.01
Other	1,119.64

The quantity of radium in each form, as shown by the reports, is as follows:

	Milligrams
Applicators	4,186.60
Needles	42,369.63
Tubes	35,223.79
Solution	26,027.65

From the reports it is estimated that the number of patients treated annually with radium is approximately 80,000. The accompanying map shows the distribution of radium for medical use by States. Seven hundred and ten individuals, companies, and hospitals, owning 124.7 grams of radium, estimate that they need 117.4 grams more.

It has been suggested that the amount needed might be estimated from the number of persons who could be benefitted by radium treatment. As any statistics for such an estimate are very limited, it seems best to consider only malignant new growths and to neglect the 25 or 30 other conditions that, at least in selected cases, are benefitted by radium.

In 1900 when the registration area was first formed, the crude death rate from cancer was 63 per 100,000 population. In 1920 it was 83.4, and in 1929 (the latest available figures) it was 96.1, an increase over the crude death rate of 1900 of nearly 52½ per cent. In 1929 the total number of deaths from cancer was 111,569. This makes cancer the second most important cause of death. Heart disease alone with 245,000 deaths claimed a greater number of victims. One of the most striking increases in the death rate has been in the so-called external forms of cancer, such as cancer of the breast and cancer of the mouth, in which, because of the superficial position, errors in diagnosis are low as compared with the possibility for error in deep-seated cancer such as that of the stomach or other internal organs.[13] Hess, using these cancer mortality statistics as a basis, estimates that "at least 10 times as much radium could be used advantageously as seems to be held in this country at present."

13 - The above figures are quoted from Health News, issued by the United States Public Health Service on October 13, 1931.

SUMMARY

Radium was first produced commercially from residues of uranium from the mines of Joachimsthal, Bohemia. Radium-producing ores have been found in about 10 countries.

The United States Bureau of Mines devised methods of recovering radium which greatly reduced the cost. The plant operated by the Bureau of Mines from June, 1914, to January, 1917, produced about 8½ grams of radium.

Up to 1926, 202½ grams of radium have been produced in the United States;none has been produced since that date. 85.87 grams have been imported into the United States, chiefly since 1923.

It is estimated that 2 grams per year, or a total of 30 grams, have been used for luminous materials.

In about 83 per cent return on over 6,600 hospitals and clinics in the United States, 287 report having 85.8 grams of radium; 128 hospitals and clinics each have 75 milligrams or more; 414 physicians report having 33,286.93 milligrams, and 171 physicians each have 75 milligrams or more; 9 laboratories and companies report having 5,545.42 milligrams, 5 of which each have 75 milligrams or more. New York State reported 29,800.61 milligrams. No radium was reported from one State and no radium was reported in hospitals from five States. Seven hundred and ten individuals, companies, and hospitals, owning 124.7 grams of radium, estimate that they need 117.4 grams more. From the reports, it is estimated that approximately 80,000 patients are treated annually with radium.

I. C. 6668 December, 1932

Department of Commerce

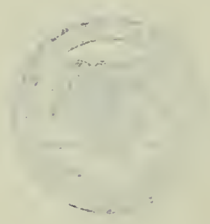
UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

PROSPECTING AND EXPLORATION FOR SAND AND GRAVEL

BY

J. R. Thoenen



I.C.6668,
December, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

PROSPECTING AND EXPLORATION FOR SAND AND GRAVEL[1]

By J. R. Thoenen[2]

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INTRODUCTION

The objective of this circular is threefold: First, to describe briefly the various modes of occurrence of sand and gravel. Second, to emphasize the need for adequate prospecting and exploration prior to the expenditure of funds for development and plant construction. Third, to describe various methods of prospecting and exploration and their application to the several modes of occurrence of sand and gravel.

Sand and gravel may be found in nearly every county or township in the United States. Commercial deposits, however, are by no means so widespread.

The introduction of concrete and the consequent impetus given to consumption of gravel and sand aggregates has seriously depleted deposits in some congested areas. Increasing restrictions in the specification requirements for marketing sands and gravels have further narrowed the economic areas.

On the other hand, the combination of increasing transportation costs and decreasing selling prices has forced the decentralization of gravel production. As a result smaller, intermittently operated, and widely scattered portable or semiportable unit plants have been encroaching on the territory formerly supplied from centrally located pits with large treatment plants.

Generally speaking, the small portable plant can not produce any great variety of materials to as exacting specifications nor as economically as the large permanent plant.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6668."

2 - Senior mining engineer, U. S. Bureau of Mines.

The imperative and insistent demand for constantly decreasing construction costs has focused attention on ways of eliminating or reducing transportation costs which make up such a large item in the consumers' bill. Substitution of autotruck haulage for railway transportation has been the solution in some localities. Improved mechanical equipment of the portable or semiportable type with which materials could be exploited nearer the point of consumption and thus effect a reduction in the cost of freight has been the answer in other localities.

To meet this demand manufacturers have made great improvements in portable equipment, and now many of the more modern wayside plants can compete on nearly equal terms for specification material with their larger adversaries within certain specification limits. This does not mean that the small plant is displacing the large plant. A portable plant is ordinarily set up to produce material to meet the specifications for a single construction project. By careful study of the deposit the plant can be designed to produce the desired material economically for that particular job and location. Upon completion the plant must be moved and redesigned for its next set-up.

The large plant, on the other hand, is designed to produce a variety of materials to meet different sets of specifications. Therefore, although it can meet consumers' demands for quantity and quality over a wide range of divergent requirements, it is handicapped by distance and freight charges in supplying certain customers situated closer to local deposits on which specially designed small plants can be established.

This decentralization of production has accentuated the need for careful prospecting and exploration. The operator of a large established plant has, over a period of years, accumulated an extensive knowledge of the physical characteristics and limits of his deposit. New deposits, not previously worked, present a problem in ascertaining their suitability for the particular project they are called on to supply. Without careful preliminary search and testing there is grave danger of expensive development and attempted exploitation of deposits which are totally unfitted for the market they were hoped to supply.

The sand and gravel operator is principally interested in the mode of occurrence of the material. The prospector, on the other hand, is interested in the mode of formation of the deposits and the origin of their constituents. Sand and gravel deposits are formed from widely divergent materials in various ways, and require vastly different methods of exploitation. Therefore any intelligent attempt at opening new property should be preceded by the most carefully planned exploration as a foundation for proper development and plant design. This requires a knowledge of the origin of the materials that make up the deposit, their method of collection and deposition, and the various ways of testing virgin areas, in addition to a thorough knowledge of the proper type of production equipment best suited to local conditions of deposit and market.

PRIMARY FORMATION OF ROCKS

Rocks as originally formed are of igneous (molten) or sedimentary (deposited) sources. Examples of igneous rocks are trap, granite, lava, etc. Examples of sedimentary rocks are limestone, sandstone, shale, etc. Sand and gravel are the disintegration products of both igneous and sedimentary rocks.

The first rock formations to appear on the earth's crust were eruptive or igneous. These solidified from molten magmas by cooling. As they solidified or froze they were at once subjected to the disintegrating effect of the atmosphere and recurrent changes in temperature.

Frequent and wide variations in temperature with their expansive and contractive effects may themselves break up rock masses. It must be remembered that all igneous rocks are com-

posed of various assortments of minerals. Heat and cold affect different minerals in different degrees. Therefore alternate heat and cold will cause internal stresses within the rock mass until finally planes of weakness develop and fractures occur along them. These original fractures may be minute in size but sufficient to allow the entrance of air and its burden of moisture. As the moisture-laden air penetrates these fractures or cracks, the attendant decreases in temperature condenses the moisture and leaves it behind. This illustrates one way in which moisture may penetrate a rock mass.

Moisture may also penetrate a rock mass through capillary action along the crystal faces of the constituent minerals.

A third method is by direct solution. Igneous rocks are ordinarily considered insoluble. However, there are practically no known minerals or rocks that will not dissolve in water under proper conditions of temperature, pressure, and time. Certain mineral constituents of rock masses react more readily to solution than others; hence, their presence offers points of differential attack by direct solution.

Irrespective of the method of penetration, the presence of water within a rock mass is a most potent agent of its destruction and disintegration. Speaking broadly, water is a better solvent or penetrating agent the higher its temperature. In areas where atmospheric temperature changes range from above to below freezing, the action of frost greatly accelerates the disruptive effect of moisture. As water freezes it expands, and when confined as in rock masses it builds up pressures to 150 pounds per square inch.[3] This internal force is capable of disrupting the strongest rocks.

Some minerals take up or absorb water by hydration. By so doing they increase their volume and thus set up internal stresses similar to frost action. Other minerals oxidize from contact with the atmosphere and thereby their cementing and cohesive strength is weakened.

In these ways the original igneous rocks were gradually broken down to masses ranging in size from fine particles to huge bowlders.

Another process of disintegration was the movement of these fragments from their residual position by stream and gravity action. As the streams pushed and rolled them the constant abrasion between individual pieces further reduced their size by detaching coarse or fine particles. Thus were formed gravel, sand, and silt, which were carried by wind or water and deposited in flood plains, and in the beds of streams, lakes, or oceans. This sand and silt may remain unconsolidated, or under the pressure of the sea and by the aid of dissolved reagents in the water may be consolidated into the second class, or sedimentary rocks.

The preceding description of the elementary principles of early geology presents the essential differences in the formation of igneous and sedimentary rocks.

Further shrinkage of the earth's crust due to cooling set up enormous stresses within the earth which caused displacement of the surface and the formation of mountain ranges or lesser uplifts of land masses. (Geologists are not in complete accord as to the details of these mountain-building forces.) This movement brought portions of the sea bottom above the water line, while at other points land masses sank below the sea. As a result, sedimentary rocks were elevated to positions from which they in turn were subjected to erosion by air and water. Conversely, igneous rocks were submerged to new sea bottoms and thus became the floor on which new formations of sedimentaries were deposited.

At times this shrinkage of the crust caused surface movement beyond its tensile strength. This resulted in breaks in the crust and the relief of internal pressure by extrusion of molten masses. Volcanic action is typical of this phenomenon. These eruptive or extrusive

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3 - Stone, R. W., Molding Sands of Pennsylvania: Topographic and Geologic Survey of Pennsylvania, Bull. M 11, 1928.

formations covered older igneous or sedimentary rocks alike.

Visualizing a continuance of these several formational phenomena over millions of years and under temperature changes unknown to-day presents a picture of the ways in which the present surface became such a mixture of mineralogic, geologic, and structural complexities.

OCCURRENCE OF SAND AND GRAVEL

To be of commercial value sand and gravel deposits must contain sufficient tonnage to warrant the erection of a recovery or treatment plant. Such deposits have been formed in a number of ways. Some fine sands have been picked up, transported, and deposited by wind. These are found in deserts and on some lake or ocean beaches. As a rule, they are too fine for commercial purposes. Except for these eolian deposits, all sand and gravel have been collected and concentrated by water or ice.

Roughly, commercial sand and gravel deposits may be classified in four groups with reference to their method of formation, as follows:

1. Residual.
2. Fluvial.
3. Marine and lake.
4. Glacial.

Residual Deposits

Sand and gravel found in situ or at the same location as the parent rock and resulting from its weathering or atmospheric disintegration are called residual deposits.

They are commonly unstratified, heterogeneous mixtures of bowlders, pebbles, sand, and clay. They may contain much soft or inferior material commonly called rotten stone. This soft material, although originally dense and hard, has through prolonged weathering become porous and weak in structure.

Such materials have not been subjected to the abrasive action of transportation by water or ice. The particles may be angular or rounded. In either case, the surfaces will be rough and granular as distinguished from the smooth surface of water-worn material.

Residual deposits commonly form a rock mantle over the parent formation. They are ordinarily so intermixed with clay as to be of little commercial importance. In some localities, however, when other forms of deposit are absent residual deposits have been and are being utilized.

Fluvial Deposits

Sand and gravel picked up and transported by stream action form fluvial deposits.

Such deposits may be more or less stratified and they frequently show rough sorting in sizes. Coarse sand and gravel may be interspersed with lenses of fine sand or clay. Deposits have their long axis parallel to the direction of the stream flow. The beds usually vary greatly in thickness and may be complex in composition.

Particles are as a rule poorly sorted and may be angular or rounded. When rounded the surfaces are typically smooth. Whether they are angular or rounded depends on the hardness of the rock and the distance the material has been transported. The less resistant rocks such as limestone, shale, or sandstone will form rounded pebbles in comparatively short distances while the harder materials such as flint and quartz may travel a considerable distance before losing their angularity.

Fluvial deposits usually overlie an eroded rock floor, but may be found above finer materials. Since the transporting medium is water they are found in a variety of structural forms. Present rivers are constantly constructing, tearing down, and reworking bars within their banks. Similar deposits have resulted from similar action by ancient rivers which have long since cut down to lower levels, been diverted to other courses or have dried up. During periods of high water rivers leave their banks and extend over wide flood plains leaving considerable material behind as they recede. At their mouths they deposit their loads in delta formations. Swift mountain streams emerging from narrow canyons or gorges onto broad flat plains deposit their material in alluvial cones. Old streams in wide valleys leave teraces behind as they cut their way down to lower levels. These are all typical fluvial deposits.

Marine and Lake Deposits

As the term implies, marine and lake deposits are formed in ocean or lake beds or beaches. Subsequent to their deposition they have been uplifted from the sea to form continental land surfaces or the lakes have been drained, or dried up, leaving their beds exposed.

In structure these deposits frequently show well-sorted materials with coarse and fine particles segregated. The various segregations usually have their long axes parallel to the shore line. Such deposits are rarely over 100 feet thick.

Individual particles of marine sands range from angular to well rounded, while pebbles are usually well rounded and smooth.

Marine deposits consist ordinarily of hard, tough materials received from streams and reworked by wave, tidal, and marine-current action. Occasionally pebbles of limestone of organic origin, such as corals, may be present. Due to the erosion of shore lines and subsequent continental uplift, deposits of marine sand and gravel are frequently found on the landward side above consolidated rocks of igneous or sedimentary origin. Subsequent erosion by streams may leave marine sands and gravel at the tops or on the slopes of interstream areas or hills. Erosion by the sea or lake may leave terraces parallel to the shore line.

Sand and gravel collected in lake beds present structures similar to marine deposits except that the materials are usually not so well sorted and cleaned. They are also less apt to be well rounded.

Glacial Deposits

Glacial deposits of sand and gravel are found along the Canadian border in the western States, in the north-central or Great Lakes area, and in the northeastern portion of the United States. They are confined to that area which was covered by the great ice sheets that crept down from Canada and to the mountains of the West where isolated local glaciers occurred.

Structurally the materials are largely unstratified heterogeneous accumulations of bowlders, gravel, sand, and clay. These accumulations are left as ridges or irregular deposits termed "moraines." As the ice from local mountain glaciers proceeds down the valley, talus from the mountain sides falls upon it and is carried on. The glacier by pressure and abrasion also picks up and carries material along in its base. When the melting rate at the terminus equals the rate of advance of the ice, the front remains stationary. The accumulated debris is then deposited in great terminal or marginal moraines marking the temporary edge of the ice. As the ice retreats, the debris is left as lateral moraines along the valley sides and ground moraines on the valley floor. Terminal moraines thus have their long axes parallel to the ice front while lateral and ground moraines usually have their long dimensions in the direction of ice flow.

Recessional moraines are terminal moraines marking points of pause in the retreat of the ice front. Later advance of the ice may totally remove these deposits or cut through them and on subsequent retreat leave other material as terraced lateral moraines.

The great ice sheets originating about Hudson Bay where igneous rocks predominate and sedimentary rocks are comparatively scarce brought down huge quantities of bowlders, sand, and gravel of igneous origin. They also scoured out valleys in sedimentary rocks en route and carried along vast quantities of this material. Therefore glacial materials are composed of a mixture of hard and soft rocks. Most of this material was carried at the base of the advancing ice where it was subjected to terrific pressure and abrasion and also acted as an abrasive on the formations passed over by the ice. There is not much stratification or sorting in moraine construction itself, but in places much material was rehandled by escaping melt water and deposited as stratified outwash in valleys leading away from the moraines. This action is described under fluviogl cial deposits. Individual particles are seldom well rounded but are usually subangular and the larger pieces frequently show striated surfaces caused by abrasion against underlying rock formations while held in the base of the moving ice. Fresh feldspar and other decomposable minerals may be present where the deposits have not been subsequently subjected to excessive weathering. Such materials have been transported in the mass of the ice where they were not subjected to excessive abrasion.

The glacial data on Fig. 1 is from an unpublished map compiled by Wm. C. Alden of the U. S. Geological Survey and shows the principal terminations of the various advances of the ice sheets.

Although the preceding methods of formation are quite different in their manner of operation and produce deposits quite dissimilar in character, there are few deposits of sand and gravel which do not present evidence of more than one formational agency.

As examples, marine deposits when elevated to continental land surfaces are subjected to erosion. Marine gravels are thus picked up, carried off, and redeposited in fluvi .l deposits. Melting gl .ciers form the sources of streams and rivers which pick up, transport, and redeposit moraine material as fluvial deposits. Likewise residual materials may be picked up by a change in the course of a stream or by an advancing glacier and redeposited as a fluvial, glacial, or even marine deposit. Consequently present sand and gravel deposits may contain materials having complex histories.

GEOGRAPHIC DISTRIBUTION OF DEPOSITS

In order to apply the preceding principles of geology to present-day conditions in the United States the sand and gravel prospector must have some knowledge of the physiography and topography of the country. A detailed description of the multiplicity of natural phenomena which brought the formation and development of continental United States to its present state is entirely without the field of this paper. For such data the reader is referred to the many published treatises on the geologic history of our country to be found in public libraries. For more detailed study he will find much of value in the reports of the U. S. Geological Survey and the various State geological surveys.

In brief, the United States may be divided physiographically into four general areas, namely:

1. The interior plains.
2. The plateau and mountain areas.
3. The Atlantic and Gulf coastal plain.
4. The glaciated area.

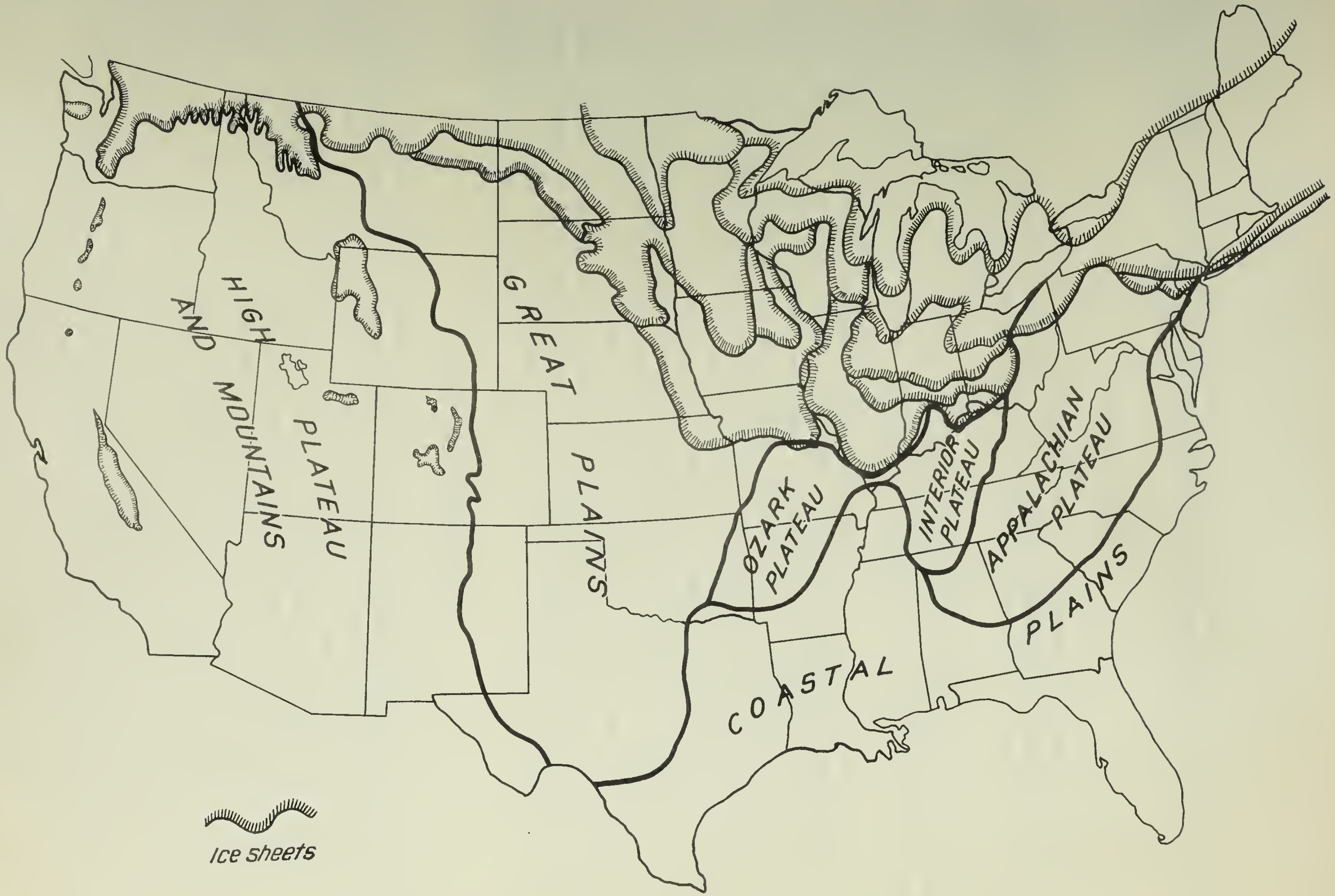

Figure 1.- Physical divisions of the United States

Figure 1 shows roughly the sections covered by these divisions. For further subdivisions the reader is referred to a map compiled by N. M. Fenneman entitled "Physical Divisions of the United States" and published by the U. S. Geological Survey in 1930.

Interior Plains

The interior plains comprise the drainage area of the great Mississippi River system, extending from Pennsylvania on the east to the Rocky Mountains on the west and from the Canadian border to the coastal plain area along the Gulf of Mexico. They are overlapped on the north by the glaciated area.

They consist of broad plains of from low to moderate relief with a few low plateaus. The principal rivers are broad, slow-moving streams and the plateaus for the most part are low, well-rounded, and soil-covered. In other words, this is an area showing the effects of long-continued weathering and surface erosion.

The area intermittently covered by ancient seas in whole or in part contains bedrock predominently sedimentary in origin. The interior plains comprise only comparatively small areas of violent crustal movement. Movement over most of the area has been slow alternating elevation and subsidence of the land surface with advance or recession of the shore line.

Local areas which have not been subjected to stream erosion and are underlain by limestone and similar rocks may and do contain deposits of residual gravel. Typical examples are the accumulations of flint nodules in a clay matrix so common in portions of the central west.

Stream erosion combined with surface weathering has collected and concentrated sand and gravel in present and ancient watercourses and their flood plains. In these areas the prospector will find his greatest source of sand and gravel in the beds and flood plains of the present rivers. However, he should not overlook the long abandoned beds, flood plains, and terraces of earlier watercourses. Reports of the U. S. Geological Survey and the State geological surveys will be of assistance to him in showing where these may be found. Geophysical methods may also be of assistance. (See p. 16.)

Plateau and Mountain Areas

Plateau and mountain areas comprise several separated sections such as the Appalachian and Ozark plateaus and the Rocky Mountains and Pacific Mountain systems enclosing the great basin and range area of the west. They are characterized by steep slopes, barren, or with light soil, on bedrock of predominently igneous origin, with swift streams in narrow valleys.

In the geologically older sections such as the Appalachian, the mountain slopes are less abrupt and the contours more rounded. At the base of the Blue Ridge Mountains and to the southeast covering the north and western portions of Georgia, South and North Carolina, and Virginia, the topography is that of a comparatively narrow plain abutting the mountains in which the erosion has not yet completely rounded the contours of the ridges. The area is a nearly flat or broadly undulating land surface produced chiefly by the work of rivers assisted by weathering. In this section the old original plain has been elevated and is now undergoing a second erosion period in which the streams are cutting deeper courses.

In the Ozark Plateau region residual gravels may be found on ridges of resistant rocks, but commercial deposits will occur in the river beds and flood plains and in alluvial fans.

In the western mountain plateaus and ranges many complex conditions exist. In places isolated mountains and short ranges are interspersed with broad desert plains. In others the ranges are continuous with narrow steep valleys.

The desert sands are seldom of commercial value except where concentrated in deposits of comparatively small area by periodic desert streams. These streams are dry the greater

portion of the year but attain torrential dimensions during cloud bursts or periods of melting mountain snows. In consequence they often bring accumulations of coarse material long distances in a single season. They also frequently cut new courses, tearing up old deposits and building new ones.

Where narrow mountain valleys debouch upon broad foothill plains the rapid, heavy-laden mountain streams are forced to deposit their burden rapidly. In consequence they form alluvial fans or cones at the valley mouths. Ordinarily the channel is not so deep but that the stream can break out at times of high water and take a new route on a lower part of the cone. When deposition has gone too far in one place there is a shift in the watercourse to a new position, so that the cone is formed symmetrically, banked up against the mountain side.

If these cones coalesce they form a piedmont alluvial plain which spreads like an apron along the front of the range.

These cones or fans are accumulations of coarse and fine material but ordinarily contain little clay or silt. The velocity of the depositing stream has been too great to allow silt to settle and it is carried beyond to the plain below. There is occasionally some grading, particularly the accumulation of bowlders at the base of the cone. These accumulations may be covered again by later finer material as the stream course changes so that the cone structure will contain irregularly placed accumulations of coarse and fine material throughout its mass.

These deposits are characteristic of the Pacific coast. They may extend for miles along the coastal range and are often hundreds of feet in vertical thickness.

In places the alluvial fans as originally formed were subsequently further eroded when the streams cut channels through them and redeposited part of the material at lower levels, leaving alluvial terraces. Such terraces are common in some of the high plateau areas where the original fans were formed by streams heading in glaciated mountain gorges and depositing glacial outwash material. With the source of this glacial material exhausted by the retreat of the ice the streams had less material to deposit and began cutting into the fan formation, redepositing it in lower fans. These latter formations are found in eastern Oregon, Washington, and southwestern Idaho.

A particularly interesting form of deposition occurs around the boundary of Great Salt Lake. This lake in former periods of geolcgic history was much larger than at present. In this earlier stage sand and gravel brought to it by mountain streams were deposited along the old shore line as beaches. As the lake was drained or evaporated these old shore lines were left as terraces and afford a plentiful supply of well-graded sand and gravel at the present time.

In the plateau and mountain regions the principal sources of commercial sand and gravel are in alluvial fans, terraces, flood plains, and river beds.

Coastal-plain Deposits

The Altantic and Gulf coasts of the United States from New York to the Mexican border have been subjected to one or more gradual uplifts whereby the land area has risen with respect to se level. Thus ancient ocean beds have been elevated to become coastal plains. These old sea beds were formed by the accumulation of gravel, sand, and silt eroded from adjacent land areas. Concentration has been accomplished by wave action, by ocean currents, and in part by the depositing streams. Considerable stratification is found, as well as complex cross-bedding. Some cementation has taken place whereby material deposited as unconsolidated sand, gravel, or silt has become sandstone, conglomerate, or shale. In other cases no consolidation has taken place and the material has been elevated in its original form of deposition.

This uplift was a very slow process extending over long periods. Previously established watercourses on the original land surface continued to function during these periods and unless the rate of uplift was too rapid they maintained their elevation with respect to sea level. In effect they cut through the rising surface as fast as it was lifted. Those areas between watercourses being unaffected by stream erosion retained their undersea character. Thus, in coastal plains we find deposits which present all the characteristics of, and are, in fact, beach sand and gravel on the tops of hills.

Periods of uplift followed by periods of quiet allowed the sea to erode the raised bed and form terraces in the terrain. Some of these older terraces in Virginia and Maryland have been classed by geologists as of fluvial origin in the form of alluvial plains. Also in southern Georgia some geologists are of the opinion that the sands occurring there as the surficial mantle are of wind-blown or eolian origin rather than marine.

Rivers within coastal-plain areas are slow moving, broad, and shallow. Except in flood periods they transport only the finer sand and silt. Material brought down during a period of high water is deposited on a mud or silt bed and covered with more mud as the current again assumes its normal flow. Thus the gravel found in the beds of coastal-plain rivers is apt to be badly interbedded with fine sand and silt or clay.

The remains of old marine deposits will be found as dry-bank gravels in areas between drainage basins. Owing to the sorting action of waves they may be more or less graded as to size.

The Glaciated Area

The Canadian ice sheets extended (see fig. 1) southward in the Great Lakes region to the southern portions of Illinois, Indiana, and Ohio, covered most of New York the northern portion of Pennsylvania, and New Jersey and all of New England. To the west they covered nearly all of Minnesota and Iowa, northern Missouri, the northeastern portion of Kansas, eastern Nebraska and the east half of South Dakota, and all but the southwest quarter of North Dakota. Further west they extended only 50 to 150 miles south of the international boundary and there were many local mountain glaciers. The southwestern quarter of Wisconsin, singularly enough, seems to have been an island untouched by the ice although completely surrounded by it. (See fig. 1.)

In advancing southward the ice sheets moved immense quantities of material, much of it from the highlands of Canada and Labrador, but the greater portion from closer sources. In this advance they scoured out the valleys now occupied by the Great Lakes, and when they melted deposited their loads in terminal and ground moraines.

Their tendency was to cut off minor irregularities in previous surface topography, smooth the hills, and fill many of the valleys, leaving in the interior a comparatively level or gently undulating terrain, but with deep wide troughs controlling the axial movement of the ice, and to leave the whole area covered by an unconsolidated mantle of sand, gravel, and silt.

The glaciofluvial waters from the melting ice followed preexisting watercourses from the edge of the ice or formed new courses for themselves. Originally the direction of flow of some of these preglacial rivers may have been northward, but this direction was blocked by the ice and the flow reversed or diverted.

Subsequently these glaciofluvial waters cut through and began tearing down the glacial moraine deposits. Material picked up in this way was deposited later as outwash plains and terraces.

Glacial waters also formed another type of deposit. Ice melting on the surface or beneath the glacier formed streams which in turn melted more ice which not only augmented their

volume but cut channels through, in, and under the body of the glacier, liberating and picking up sand, gravel, and clay enroute. These streams emerging from the face of the retreating ice deposited their load as they fell. Some deposits were made by streams flowing in ice-walled channels or tunnels. As the ice melted the material was left as low, rounded ridges with their long axes parallel to the direction of ice movement. These are termed eskers.

The whole glacial area was thus covered by a mantle of unconsolidated drift, with deposits of sand and gravel, here and there especially along the valleys. Between the terminal moraine ridges are found broad, smoother tracts with low hills and some knolls, or ridges. These are ground moraines and eskers.

Taken as a whole these several types of deposits comprise a heterogeneous collection of sand, gravel, bowlders, and clay.

Bordering some terminal moraines are broad, gently sloping, outwash plains of fairly well assorted sand and gravel. These usually front on terminal moraines and were formed as outwash plains by glacial water. Terminal moraines left by the retreating ice front dammed valleys and impounded watercourses to form the countless lakes characteristic of the topography of Michigan, Wisconsin, Minnesota, northern Pennsylvania, New York, and the New England States. The streams from the melting ice deposited much sand and gravel along their valleys where this material is now generally found in the form of terraces.

Outwash plains are typical of the deposits of Illinois, Indiana, and Ohio.

In South Dakota all the sands and gravels east of the Missouri River were brought by ice or glaciofluvial waters.

In Pennsylvania the glaciers brought rock material from Canada and New York State, and left it in moraines and glaciofluvial deposits.

The sand and gravel prospector has no problem of discovery for his material in this region. His problem here becomes one of search for quality. His object is to find the reworked or glaciofluvial deposits in preference to the moraines and eskers. However, the latter should by no means be overlooked, for their silt or clay content may be in such form as to afford little difficulty in its removal.

The beds of streams and some lake beaches and terraces afford promising sources for clean, sorted materials.

PROSPECTING

Before proceeding with the discussion of prospecting the following definitions are made in order that there be no misunderstanding of the meaning of terms used.

Prospecting is herein defined as the act of searching for, with the object of discovering, deposits of mineral.

Exploration is defined as the act of examining a deposit when found, with the object of determining roughly its size, shape, and value.

Development is defined as the process of preparing a deposit for exploitation, with the object of recovering its contents for sale or use.

The widespread and plentiful distribution of sand and gravel deposits coupled with the early laxity of specification requirements when the industry was young, rendered prospecting for those materials unnecessary. Discovery was a common occurrence through the agency of other unrelated activities such as the building of roads, railway rights of way, or canals, digging cellars and wells for houses, and drilling water, oil, and salt wells. As cosmopolitan communities grew, sand and gravel plants were erected to supply their markets. Prospecting for these deposits constituted the simple practice of examining existing evidence in the immediate neighborhood. With the advent of concrete and the increased demand for

aggregates, more and larger plants were erected. Consumption, however, remained in a more or less centralized market. Building construction centered around cosmopoltan areas, road construction radiated from those areas, and ballast was delivered to the railroads within their limits.

Construction, roads, and ballast made heavy demands on these original deposits and in many places depletion was serious. As knowledge of the art of making concrete grew, specifications became more and more rigid. Deposits previously suitable to market demands were found to require complicated treatment to meet changing specifications. In consequence, many deposits were abandoned as not containing sufficient material in reserve to warrant the expense of an elaborate treatment plant. This caused operators to cast around for new deposits. Many were opened and expensive plants were erected only to fail because the material proved unsuitable for the market. Other failures resulted because the operators erected plants on the new deposits identical in design with those just abandoned without realizing that the material in the new pit would require entirely different treatment. Many such plants erected by individuals were underfinanced and when it became apparent they were unsuitable the operator failed because of lack of funds to make the required alterations in design. New investors, many times with no knowledge of the business of producing sand and gravel other than that some men had made a profit from it, stepped in and bought these new plants at a reduced valuation. Some were successful after redesign, others were not. The fact remained, however, that deposits of sand and gravel were known to exist at these points and that plants were standing ready to operate. Such plants, to the initiate, are monuments of failure, but are intriguing to the novice as offering opportunity for rehabilitation and investment of idle capital.

Many of these failures could have been prevented if previous study had been given by adequate prospecting for the deposits and exploration when found.

Recently conditions have again changed. Railroads hampered by lack of earnings have radically curtailed ballast purchases. Building construction has fallen off badly. The gravel producer has been forced to concentrate his sales effort on highway construction. Here also conditions have altered. Where formerly highway construction centered and radiated from community centers the demand is now for intercommunity roads. This has created a market for aggregates at widely separated consuming points outside of metropolitan areas. In other words, the aggregate market has been decentralized.

Freight rates or transportation expense, always a large item in the consumer's bill, became of still greater importance as delivery points receded from old production centers. Aggravating this condition, increasing freight rates forced many operators to revert to auto-truck delivery over paved highways to new projects.

This condition afforded an excellent opportunity for the opening of new deposits favorably situated, with respect to truck or rail haul, to this scattered highway construction. Again many new plants were built. Rigid and differing specifications requiring treatment plants of elaborate design coupled with an unprecedented growth of consumption and the inertia of thought respecting radical departure in design, caused operators to follow previous precepts in the design of these new plants. In consequence many were built near the site of new road construction after plans entirely too elaborate and for much too great a tonnage. Upon completion of the particular road project which instigated their erection their owners found themselves with deposits of material and plants to treat them but without local markets. This forced them to attempt to extend their field by reducing prices. Thus was ushered in a period, of intensive, and many times destructive, competition.

Again, had better forethought been expended in prospecting and exploration in conjunction with a more far-seeing study of markets there would have been less liability for failure and overcapacity construction.

At present there is a decided movement toward decentralization of production to meet decentralization of consumption. This is exemplified by the growth in the number of small wayside pits erected by new producers, by highway contractors to supply their own jobs, and more recently by operators of large centrally located plants to compete with the two previous types in an effort to hold their market territory.

This trend has resulted in the improved design of small portable or semi-portable plants. These are being erected with the sole purpose of supplying material for a certain construction project in which a material of definite specifications is required. These plants are designed to produce that material primarily and it is not intended that they should be capable of supplying material to meet a multiplicity of specifications.

The opening of these new deposits for more or less temporary operation requires first a careful search for locations within economic or competitive shipping distance of the project they are intended to supply, and second, a careful exploration to determine their size, and suitability. There are practically no sand and gravel deposits in existence to-day capable of supplying any single construction specification without some form of treatment. Size gradations of natural deposits must be altered to meet engineers' specifications. This means the elimination of certain sizes and a recombination of others. These eliminated sizes must in many cases be wasted. In this the small plant supplying a single project is at a disadvantage because it has no outlet for these wasted materials. Soft particles must be removed, as must clay and silt also. This requires washing and at times hand picking. Washing requires suitable supplies of water. The type and design of the production equipment will depend upon the physical characteristics of the deposit. The design of the treatment equipment will depend equally upon the physical character of the material in the deposit and the gradation and specification desired. The screen sizes must be correlated to produce the desired gradation from the raw or washed material. The method of washing must be suited to the quantity of material that must be removed. Assurance must be had that the water supply is adequate and suitable.

All of these items of study are based on knowledge which can be and should be obtained by a careful program of prospecting and exploration. No plant should be erected without this preliminary search and study. The extent of the prospecting and exploration campaign will necessarily be adjusted to the size and complicity of the market it is desired to supply.

Prospecting Methods

Prospecting for sand and gravel is usually done by one of two general methods: The direct method, wherein the prospector sets out to find a visible deposit; and the indirect method, in which he relies upon topographic or geologic knowledge to guide him to likely localities.

In the direct method the prospector endeavors to find a deposit which outcrops above the soil or water level, is visible in clear shallow water, can be seen in railroad or highway cuts, or is known to exist from data collected in the digging of cellars, wells, or post holes. This method involves a great deal of time in the observation and search for visible outcrops and in the collection of excavation information. A large percentage of the results will be negative. In such a search the prospector must systematically examine the whole area.

The indirect method presupposes a knowledge of the structural geology of the region under investigation. This method is primarily one of elimination in which the prospector avoids all that area which has unfavorable topographic or geologic structure. He thereby confines his direct examination to only that area which his knowledge indicates may contain commercial material. His first procedure in this method is to collect all available data on the geology and physiography of the territory. The published reports of Federal and State

geological surveys will be of considerable assistance at this stage. These reports need not necessarily deal with sand and gravel resources only, but may relate entirely to geologic structure. The interpretation of the causes and results of the local structural history will furnish the key to likely sources. Upon locating favorable structures a knowledge of the genesis of the formation may give valuable information as to the character and value of the deposit itself. In such a search much of the prospector's time and money are saved by his knowledge of where to search without the necessity of examining the whole region.

Table 1 shows the principal guides to surficial examination for sand and gravel deposits.

Supplementing and in further explanation of the following table the following comments are in order.

Topographic Features

Topographically the earth's land surface may be divided into two general divisions: One of high relief or mountainous areas, and one of low relief or broad valleys and plains. In the first, erosion is active and products of erosion are rapidly removed with comparatively little deposition occurring. In the second, the erosion products of the first are deposited and erosion is very slow. The second division may be further subdivided into areas in which the material has been deposited by transporting agencies such as ice or water and others where material is left in place by weathering.

Therefore, regions of high relief being deficient in conditions favorable to deposition are not as promising territory for the prospector as those of low relief. This, of course, is a general statement and must not be taken as covering all conditions. For instance, swift mountain streams may debouch from narrow valleys into broad piedmont plains with suddenly decreased slope. This causes deposition as alluvial fans and plains which may be formed in mountainous regions, only to be later reworked by the continuing stream action and tansported to and deposited on lower plains.

Low, rounded hills in glacial regions suggest moraines or eskers, depending on the relation of their long axes to ice movement. Rounded ridges outside of glacial areas suggest residual deposits.

Gently sloping plains at the base of mountain ranges suggest alluvial fans or plains. Similar plains not contiguous to mountains but in the vicinity of glacial territory suggest outwash or fluvioglacial plains. Flat-topped plateaus in coastal-plain areas suggest marine deposition.

Rivers with swift currents and narrow valleys indicate small bar deposits of coarse gravel and bowlders. Narrow, moderately swift streams in wide valleys indicate gravel and sand bars in their beds and terraced valley walls. Broad, slow-moving rivers traversing flat or gently sloping plains indicate fine gravel with accumulations of sand and silt.

The grade of a drainage system will vary from steep slopes in the mountains to zero in the valleys. The transporting power of a stream is dependent on the velocity of the current, which in turn is dependent on the grade and character of the stream bed, the height, character, and alignment of the banks, the depth and volume of water flowing, and the amount of material being transported. Old publications state that the size of a particle which can be transported by a stream varies as the fifth power of the velocity. Van Wagenen[4] gives the following table:

4 - Van Wagenen, T. F., Manual of Hydraulic Mining: D. Van Nostrand, N. Y., 1880, p. 88.

Table 1.- Key to sand and gravel deposits

Formation	Sites of deposition	Structure	Fine material	Coarse material	Associated material	Relations to underlying material
Residual	Regions of disintegration. Tops of low rounded ridges between drainage basins and outside glacial area.	No true bedding. Angular or spheroidal weathering. Heterogenous mixture.	Disintegration sands and clays such as feldspathic clay from granite.	Well-rounded (spheroidal weathering) to angular particles with rough or pitted surfaces. Flint nodules from limestones.	Associated with the parent rock because there has been no transportation.	Bowlders of weathering grade down into the unweathered rock. Differentially weathered deposits lie upon roughened surface of the parent rock.
Fluvial	Alluvial cones and piedmont plains.	Varied texture. Sand lenses common; often cross-bedded. Elongate parallel to stream flow.	Poorly sorted. Angular to rounded grains. More apt to be rounded.	All sizes up to several tons in weight, but mostly less than 6 inches in diameter. Subangular to well rounded. Smooth surfaces.	Sands and sandstones, Clay seams.	May lie upon eroded rock floor. Sometimes upon finer sediments.
	Terraces	Material accumulated in discontinuous layers. Beds vary in thickness. Sorting may be poor				
	Stream channels	Matrix may be so abundant that pebbles do not touch one another.				
	Flood plains	May be hundreds of feet thick in places.				
	Deltas					
Marine and lake	Nearly all high on beach. Hilltops between drainage basins in coastal plains	Pebbles usually touch one another. Well sorted. Possibly definite arrangement. May be pell-mell structure. Sand lenses elongate parallel to shore line. Rarely over 100 feet thick	Clean sands fairly well sorted. Angular to rounded.	Well rounded, smooth with dull polish.	Sand and sandstone.	May rest unconformably on wave-scoured rock.
Glacial	Area originally covered by the ice	No true bedding. Isolated nests and small beds of sand. Unstratified mass of miscellaneous unsorted rock	Angular. Decomposable minerals such as feldspar may be present. Rock flour containing sand grains may occur	Angular or subangular striated surfaces. All sizes.	May be associated with water-washed materials.	If unconsolidated the underlying materials are apt to be disrupted and contorted. If rock, it may be striated and grooved.
Glaciofluvial	Valley trains; outwash plains; eskers.	Similar to fluvial alluvial cones. Stratification good, poor or absent. Structure often pell-mell.	Poorly sorted; angular and subangular predominate. Feldspar may be present.	Typically subangular. Large striated bowlders.	Sand or sandstone.	Glacial abrasion on bedrock below. May rest unconformably on other deposits.

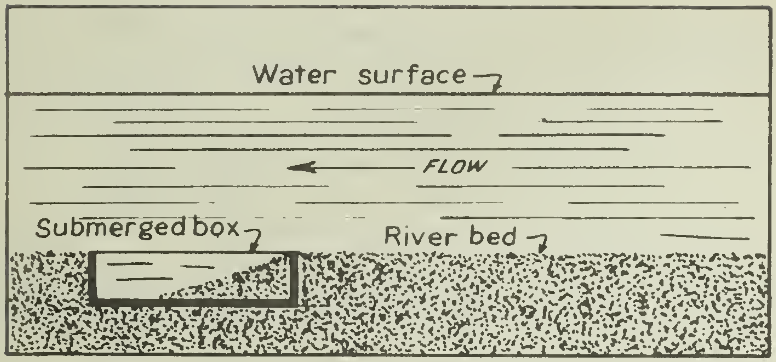

Figure 2.— Box set to test stream carrying capacity

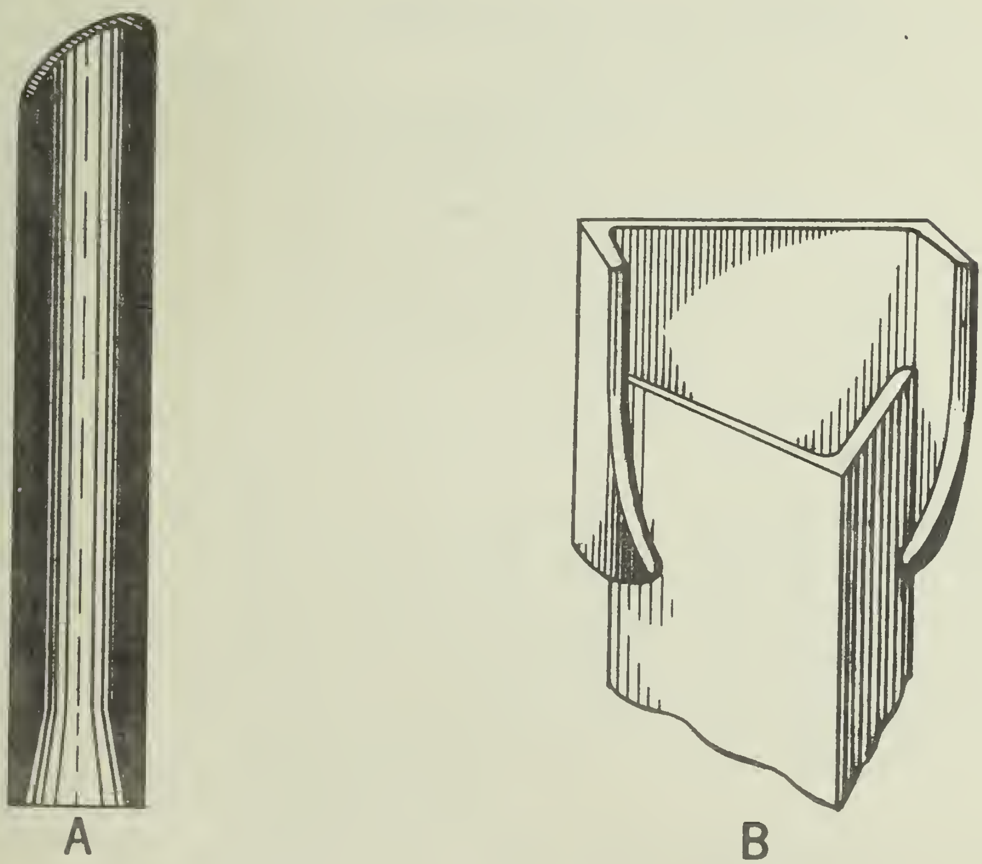

Figure 3.— A, Section of test-pipe bit; B, angle and channel

Table 2.- Transporting power of water in sluiceways

Velocity, f.p.m.	Material moved
16	Begins to wear away fine clay.
30	Just lifts fine sand.
39	Lifts sand as coarse as linseed.
45	Moves fine gravel.
120	Moves 1-inch pebbles.
200	Moves pebbles as large as eggs.
300	Moves bowlders 3 to 4 inches in diameter.
400	Moves bowlders 6 to 8 inches in diameter.
600	Moves bowlders 12 to 18 inches in diameter.

Owing to the differences in influential factors the carrying power of water in streams will vary considerably and sluice experiments are not truly comparative. A similar table is quoted by Geikie[5] as follows:

Table 3.- Transporting power of river currents

Velocity, f.p.m.	Material moved
15	Will just begin to work on fine clay.
30	Will lift fine sand.
40	Will lift sand as coarse as linseed.
60	Will sweep along fine gravel.
120	Will roll 1-inch pebbles.
180	Will sweep along slippery angular stones of the size of an egg.

Many times a rise of only 1 foot in a river will replace sand and gravel that required months to remove by dredging.

Very shoal or very swift water is not conducive of forming river bed deposits. Deep pools above or below shallow rapids are potential sites of deposits. The inside of curves in the river's course, the point of confluence of small tributaries with the larger stream, or the site of any other cause which tends to decrease the current flow indicates deposition.

A common method of testing the rate of deposition in a stream bed is to submerge a wooden box in the river bottom (see fig. 2) so that the upper edges are just flush with the bed. Observation of the time required to fill the box or preferably measurement of the amount caught in a given time calculated to the ratio between the width of the box and the width of the stream will give the information desired.

Gravel beds free from clay and silt offer favorable conditions for the free migration of underground water. Therefore in arid regions the presence of unusual vegitation often indicates the presence of underlying sands or gravels. As an example, in Georgia[6] sand

5 - Geikie, A., Class-book of Geology: Macmillan & Co., London, 1902.

6 - Teas, L. P., Preliminary Report on the Sand and Gravel Deposits of Georgia: Bull. 37, Geological Survey of Georgia, 1921, p. 132.

deposits along the fall line are easily located by the growth of scrubby, sharp-leafed, black jack pines in an otherwise barren surface landscape. In other sections, such as parts of Texas, the presence of willow trees are indicative of subsurface water in gravel occupying stream beds long since abandoned by the present watercourses.

The prospector will do well, then, to observe first the topography and physiographic features of the surface, including vegetation where it may be indicative. His next interest is the physical character of the material exposed to view. He must then correlate these data with topographic information.

The composition, shape, size, and surface markings of sand and gravel particles are important in defining the origin and estimating the value of deposits. The presence of decomposable minerals indicates short, rapid transit from source to deposit without extended erosion. Smooth-surfaced, angular to rounded pebbles indicate alluvial deposits. Pitted or rough-surfaced gravels indicate residual deposits. Angular to rounded particles with striated or grooved surfaces point to glacial action. Subangular to well-rounded gravels with smooth surfaces having a dull polish indicate marine deposition.

Geophysical Prospecting

So far as the writer is aware geophysical methods of prospecting for gravel deposits have not as yet been used by operators. During the summer of 1931 the Illinois State Geological Survey conducted some experiments with the relative resistivity method in locating gravel. This work was conducted by M. King Hubbert, Instructor in Geophysics, Columbia University, and the results were published as Technical Publication No. 463 by the American Institute of Mining and Metallurgical Engineers, 29 West 39th Street, New York.

One test made to locate the depth of bedrock below the surface gave negative results and the depth could not be determined. Unexpectedly, however, differences in resistivity were marked at two stations, and this information combined with a knowledge of the local geology indicated the presence of a sharply defined gravel deposit, probably in the form of an ancient river channel.

In another test one station showed an abnormally high value of apparent specific resistivity indicating a shallow highly resistant body underlain by a relatively better conductor. This again was interpreted as gravel by applying a knowledge of local geology.

In order to check these results, experiments were made upon land known to contain gravel previously outlined by test pits and wells. The resistivity test coincided almost exactly with the known gravel area. For details of these tests the reader is referred to the previously noted publication.

From these results it would seem that geophysical methods have a place in the field of gravel prospecting, especially in the location of deposits otherwise hidden from surface examination.

EXPLORATION

The impression that the work of the prospector ends with discovery of a promising deposit is unfortunately far too prevalent. On the contrary, with such discovery the work of the prospector has only begun. His job from here on is to demonstrate that his discovery has a commercial value. This involves a careful examination of the surface exposures, followed by a similar examination of the material below the soil mantle or clay overburden. The size or extent of the deposit is important not only to insure sufficient material to supply the market but to ascertain what possibility there may be for competitors to locate on and operate portions of the deposit beyond the property leased or owned by the operator.

Of equal or greater importance, however, is a careful sampling and testing of the material to ascertain thoroughly and accurately its natural size gradation and physical characteristics.

If exploration is thorough, the prospector will have obtained the necesary data on which the operator may pass intelligent judgment on such questions as whether the deposit is sufficiently large to operate, whether it contains the necessary materials to fill specifications for markets within shipping distance, whether there is sufficient material available, how much material must be wasted and the locations of the best sites for waste dumps, and, in addition, the type of equipment best suited for its exploitation, the best method of attack, the proper location for the treatment plant, and the type of equipment needed in preparing the material for market.

None of these questions can be answered definitely by prospecting alone. Therefore the importance of exploration and its interrelation between prospecting and development is established and emphasized.

Prior to a discussion of methods of exploration, it seems advisable to define certain terms used in connection with sand and gravel production. Authorities differ in their definitions of the terms "clay," "silt," "sand," "gravel," "pebbles," "cobbles," and "bowlders." As a matter of comparison some of these definitions are quoted herewith:

> Sand is a mass of small rock fragments in an incoherent condition.
> Gravel and coarse sand are synonymous.[7]
> Sand may be defined as a naturally occurring incoherent, granular rock material less than 1/4 inch and more than 1/200 inch in diameter.
> Gravel, similar except as to size.
> Silt, similar except as to size.[8]
> Sand is an incoherent material made up of particles of crushed or worn rock.
> Silt, similar to sand except in size.
> Clay, similar to sand except in size.
> Gravel is an incoherent rounded to subangular rock material larger than 1/4 and less than 4 to 5 inches in diameter.
> Cobbles, similar to gravel except in size.
> Bowlders, similar to gravel except in size.[9]
> Sand in a commercial sense somewhat loosely means a material consisting of rock particles which are larger than 1/200 inch in diameter and smaller than 1/4 inch.

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7 - Condra, G. E., The Sand and Gravel Resources and Industries of Nebraska: Nebraska Geol. Survey, vol. 3, pt. 1, 1911, p. 16.

8 - Martens, H. J. C., Sand and Gravel Deposits of Florida: 19th Ann. Rept., Florida Geol. Survey, 1926-27, 1928, p. 39.

9 - Teas, L. P., Preliminary Report on the Sand and Gravel Deposits of Georgia: Geol. Survey Georgia, Bull. 37, 1921, p. 1.

Gravel consists of somewhat rounded rock fragments larger than 1/4 inch and smaller than 2 to 3 feet in diameter.[10]

Wentworth makes a further size classification as follows:

	Diameter in inches Plus	Minus
Clay	-	1/6500
Silt	1/6500	1/400
Sand: Very fine, Fine, Medium, Coarse, Very coarse	1/400	1/12
Gravel: Granule	1/12	1/6
Pebble	1/6	2 1/2
Cobble	2 1/2	10
Bowlder	10	-

Sand indicates any granular rock material passing 8 mesh and retained on 270 mesh, regardless of chemical composition or manner of origin.

Gravels are fragments of unconsolidated deposits greater than 1/4 inch in diameter.[11]

Nevin also makes a further size distinction as follows:

	Diameter in inches Plus	Minus
Silt	-	270 mesh
Sand: Fine	270 mesh	70 do.
Medium	70 do.	20 do.
Coarse	20 do.	8 do.
Torpedo	1/8	1/4
Gravel: No. 1	3/8 round or 1/4 square	3/4 round
No. 2	3/4 round	1-1/2 do.
No. 3	1-1/2 do.	2-3/4 do.
No. 4	2-3/4 do.	3-3/4 do.

Sand.- The fine granular material (usually less than 1/4 inch in diameter) resulting from the natural disintegration of rock, or from the crushing of friable sandstone rocks.

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10 - Wentworth, C. K., Sand and Gravel Resources of the Coastal Plain of Virginia: Virginia Geological Survey, Bull. 32, 1930, p. 119.

11 - Nevin, Chas. M. The Sand and Gravel Resources of New York, Bull. 282 New York State Museum, 1929, p. 7.

Note.- When used without a qualifying adjective, the term "sand" is generally understood to mean the product of the natural disintegration of siliceous or calcareous rock. Sand should be distinguished from screenings, gravel, etc. The size of the particle and other physical characteristics should be taken care of in specifications. The fine material resulting from the crushing of blast furnace slag is known as "slag sand."[12]

Merrill[13] makes the following size classification:

	Diameter, millimeters
Gravel	About 2
Fine gravel	2 to 1
Coarse sand	1 to 0.5
Medium sand	0.50 to 0.25
Fine sand	.25 to .10
Very fine sand	.10 to .05
Silt	.05 to .01
Fine silt	.010 to .005
Clay	.005 to .0001

Silt.- The deposit of mud or fine earth from running or standing water.

Sand.- The finely divided material, generally of a siliceous nature resulting from the reduction of rock by natural forces to the size included under fine aggregate (minus 4 mesh).

Gravel.- Loose material containing particles larger than sand resulting from natural crushing and erosion of rocks.

Plum stone.- Same as gravel except ranging from 3 inches in diameter to 100 pounds in weight.

Cyclopean stone.- Same as plum stone except over 100 pounds in weight.[14]

These typical examples of both definitive phraseology and size present rather wide differences. The author presents the following definitions as embracing the concensus of opinion and use by the producers and consumers of sand and gravel:

Clay and silt.- The unconsolidated material, finer than 200 mesh, resulting from the natural disintegration of rocks.

Sand.- The unconsolidated granular material coarser than 200 mesh and finer than 1/4 inch resulting from the natural disintegration of rocks.

Gravel.- The unconsolidated material coarser than 1/4 inch and finer than 3½ inches resulting from the natural disintegration of rocks.

12 - American Society for Testing Materials, Standard C58-28. Definition of sand by Committee E-8: Standards, pt. 2, 1930, p. 165.

13 - The Educational Series of Rock Specimens Collected and Distributed by the United States Geological Survey: U. S. Geol. Survey Bull. 150, 1898, Chapter by Merrill, G. P., p. 380.

14 - Committee on Nomenclature, American Concrete Institute Proceedings, 1923, p. 319.

Pebbles.- Identical with gravel.

Cobbles.- The unconsolidated material larger than 3½ inches and smaller than 10 inches in diameter resulting from the natural disintegration of rocks.

Bowlders.- The unconsolidated material larger than 10 inches in diameter resulting from the natural disintegration of rocks.

These various size classifications have been grouped in Table 4 for facility in comparison.

Exploration Methods

Exploration means examination. The object of exploration is to determine the size, shape, and value of a previously discovered deposit. Discovery may be merely the observing of the material lifted with a post-hole digger, the excavated material from a cellar or well, or visible in railway or highway cuts or similar exposures. Determination of size, shape and value requires actual physical examination of the material over the whole deposit, or at least over the area owned or leased. This requires opening holes or pits at intervals from which samples may be collected that will represent as nearly as possible the deposit as a whole. These samples must then be carefully tested in various ways and the results charted or otherwise recorded. With such information at hand the prospector reaches the most important step of all, which is a proper and intelligent interpretation of his results.

There are various ways of determining the extent of a deposit lying below thin overburden. If the soil be light one can often outline the gravel below by means of an iron bar. By churning, the bar can be forced through the soil by hand. When it strikes gravel a distinctly different metallic sound will be heard and usually the gravel will be more difficult to penetrate than the soil. The same means is often used for testing gravel bars in rivers in which the gravel is covered by fine sand or silt. Surface material may thicken in places beyond the reach of the bar and such areas would be reported as containing no gravel, although it might be present at greater depth. A length of 3/4-inch pipe to one end of which a steel point has been welded will extend the depth beyond that which can be reached by a solid bar. This method is applicable to the location of gravel from 2 of 15 feet below the surface. In river bars pipes have been used to locate gravel 20 feet or more beneath the surface of the water. The method is of principal interest in prospecting but is useful in delineating a deposit when discovered. It is inconclusive in that it recovers no samples and is useless in thick or heavy overburden.

Some operators are successful in exploring with the use of ordinary steel pipe 2 to 2½ inches in diameter. One end of the pipe is forged or filed to a sharp edge with the bevel on the inside (fig. 3, A). A malleable-iron coupling or tee is screwed to the other end and the pipe is driven into the ground by wooden mauls or steel sledges. Short or long lengths of pipe are used, but with the latter platforms must be used to allow the men to reach the top end of the pipe in starting. Old diamond-drill rods can be used for this purpose. Ordinary pipe in short lengths with couplings of larger diameter than the pipe offers greater resistance to both driving and pulling than the drill rod with its flush joints.

The pipe is pulled, held over a wooden or steel platform, and tapped with a light hammer. As the material falls out the pipe is moved in a straight line. In this way the material brought up in the pipe is left in a continuous pile roughly representing its position in the ground. It can then be examined in whatever interval is required.

By forming the bevel of the cutting edge on the inside, the material entering the pipe is compressed as the pipe is driven. This helps to retain it when the pipe is pulled. Some prefer to enlarge the cutting end beyond the pipe diameter.

Table 4 - Size classification of rock-disintegration products by various authorities

(Diameter in inches or mesh)

Authority	Clay		Silt		Sand		Gravel		Pebbles		Cobbles		Bowlders	
	Plus	Minus	Plus	Minus	Plus	Minus	Plus	Minus	Plus	Minus	Plus	Minus	Plus	Minus
8 Martens, J. H. C.	-	-	-	200 mesh	200 mesh	1/4 inch	1/4 inch	-	-	-	-	-	-	-
9 Teas, L. P.	-	250 mesh	250 mesh	100 do	150 do	1/4 do.	1/4 do	4-5 inch	-	-	4-5 inch	10 inch	10 inch	48 inch
10 Wentworth, C. K.	-	1/6500 inch	1/6500 inch	1/400 inch	1/400 inch	1/12 do.	1/12 do	2-1/2 do	6 mesh	2-1/2 inch	2-1/2 do	10 do	10 do	-
11 Nevin, Chas. M.	-	-	-	270 mesh	270 mesh	8 mesh	1/4 do	3-3/4 do	-	-	-	-	-	-
12 American Society for Testing Materials	-	-	-	-	Not given	1/4 inch	-	-	-	-	-	-	-	-
13 Merrill, G. P.	0.0001 mm	0.005 mm.	0.005 mm.	270 do.	270 mesh	18 mesh	18 mesh	-	-	-	-	-	-	-
14 American Concrete Institute											Plum stone		Cyclopean stone	
	-	-	-	-	Not given	4 do.	4 do	3 do	-	-	3 inch	100 pounds	100 pounds	-
Thoenen, J. R.	-	200 mesh	-	200 do	200 mesh	1/4 inch	1/4 inch	3-1/2 do	Same as gravel		3-1/2 do	10 inch	10 inch	No limit

In dry deposits free from clay it is sometimes difficult to recover samples, as the gravel falls out when the pipe is pulled. Operating in material below the water table often has the same result, even with considerable clay present. In dry banks, however, with sufficient clay present to bind the core the pipe works fairly well.

In deposits containing many cobbles or bowlders the pipe is not successful.

This method at best is but a preliminary test and should in no case be considered conclusive. When intended and used as a preliminary test only, it is a valuable means of both outlining a deposit and collecting a rough qualitative sample.

Some operators use this method to determine the depth of overburden only. They pull the pipe as soon as it strikes the gravel. The author has no knowledge of any attempt to ascertain the thickness of a deposit by this means. The pipe may be used to depths of from 5 to 30 feet or more depending on local conditions.

Earth augurs or post-hole diggers are often used both to determine the thickness of overburden and to obtain samples of the gravel.

Neither is very successful in penetrating the gravel to any considerable depth but they do obtain information regarding the amount of overburden and a rough sample of the top of the gravel.

One Texas operator is reported as using a machine-driven post-hole digger cutting a hole 16 inches in diameter.[15] The machine is operated by a gasoline tractor and digs from 1 to 5 feet per minute in clay overburden. It is not successful in digging gravel nor in ground containing bowlders.

A recently patented method of obtaining subsoil cores is described in an article appearing in the March 3, 1932, issue of the Engineering News Record. The method consists of driving a 4 by 4 by 5/8 inch angle vertically into the ground at the desired point. A 7-inch (20-pound) channel is then driven alongside the angle in such a position that the channel web spans the outstanding legs of the angle. The two shapes are then pulled and the enclosed material is removed intact.

The lower end of the channel has the flanges turned in for a short distance so that it will overlap and follow the angle as it is driven. Figure 3, B illustrates this combination.

The equipment necessary consists of a portable crane, for erection and mobility, an air-pile hammer with suitable compressor, a pair of wood or steel leads set on a spread timber foundation for driving the steel shapes, and a pair of 6-sheave blocks for pulling.

When the shapes are removed from the ground they are laid down and separated by slipping the channel a few inches one way until the turned-in flanges disengage the angle. The channel can then be lifted off, leaving a core of the material passed through in the angle open for inspection or sampling.

This method encounters the same difficulty as pipe driving when used in ground containing large bowlders. It is also limited in depth to single lengths of the structural shapes. These have been successfully used to depths of 40 feet, however. A further difficulty in its use is the necessity for an air compressor and pile hammer. Possibly this might be overcome by an adaptation of churn-drill equipment to drive the shapes. In any event, the method has interesting possibilities and should not be overlooked in selecting exploration methods.

The churn drill is used by many operators with or without casing in exploring gravel deposits. By careful drilling and keeping the casing close to the bottom of the hole, fairly accurate information can be obtained. If the casing is not kept close to the bottom of the hole there is grave danger of contaminating the sample removed from the bottom with material falling into the hole from above. This danger is aggravated by the churning action of the

15 - Hyde, W E , Mining Treatment Methods and Costs at the East Texas Gravel Co 's Deposits: Information Circular 6537, Bureau of Mines, 1931, 8 pp

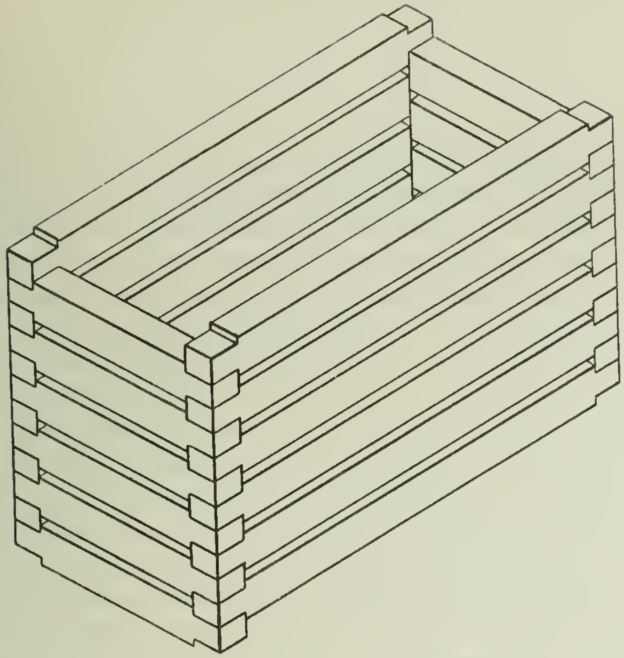

Figure 4.- Crib timbering

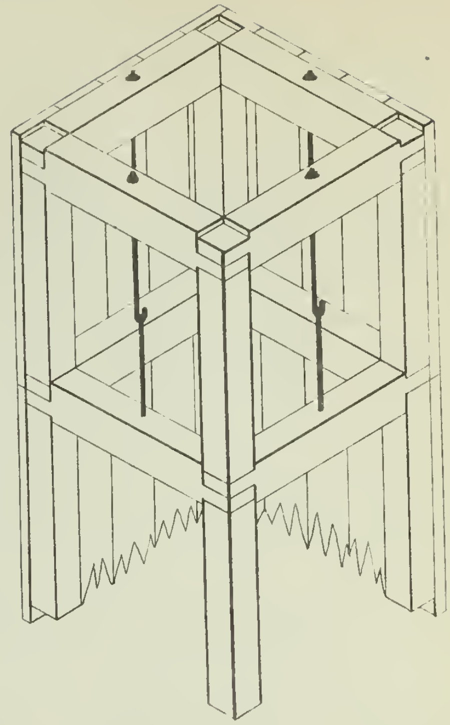

Figure 5.- Square-set timbering

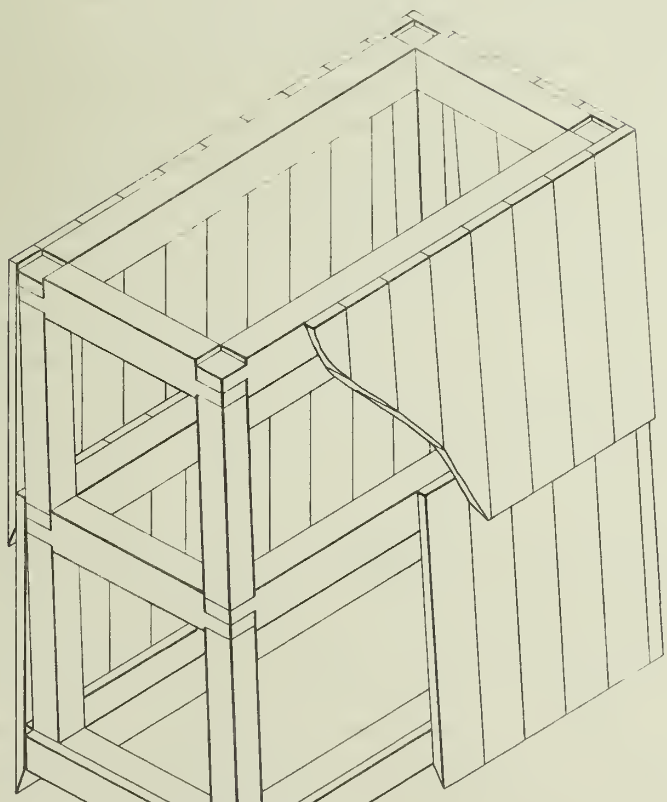

Figure 6.- Fore-pole timbering

drill itself and the whipping from side to side of the drill cable. By keeping the casing driven close to the bottom and cleaning the hole at short intervals the material brought up in the bailer will represent quite well the material passed through. Churn-drill holes in gravel should be cleaned out and the casing driven down at 2-foot intervals at most.

For exploration purposes the gasoline-driven, caterpillar-traction churn drill is perhaps the most popular type of machine. It is mobile and fairly cheap to operate. It will obtain information as to the depth of overburden and thickness and quality of gravel when the vertical thickness is not too great. It can be successfully used to depths of 50 feet or more. In some deposits where the subsurface water level is shallow, the churn drill is depended upon as the only successful means of exploring below water level. In some localities the casing is driven below the drill and the latter is used only to break up the inclosed material ahead of the bailer. This produces a more accurate sample but is not suitable for all cases because of the difficulty in driving the casing in unbroken ground.

In some ground that can be penetrated by driving casing the drill may be eliminated and a specially constructed flap valve bailer used. The lower or cutting end is fitted with a 3-pronged tempered cutting edge. Fastened to the drill cable it is churned up and down, thus breaking up the material inside the casing, and by means of the flap valve at the bottom the sample is removed by the bailer. This type of equipment may be operated by hand instead of a well drill. In this case the cable is brought up to a tripod erected over the hole and back to the ground where two to six men furnish the motive power, depending on the size of equipment used. The casing and bailer method has many advocates for sampling material below the water table, but by some operators it has been discarded in favor of the churn drill.

A modification of the casing and bailer method has been made by an equipment manufacturer. This consists of a small orange-peel bucket operating inside the casing. The bucket is operated by hand with cable and tripod the same as the bailer. As before, the casing must be driven ahead of the bucket, which is not always feasible. Where it can be used, however, this method should obtain accurate samples. As with all methods involving casing it is not successful where bowlders are encountered. Its principal field is in dry material above the water table. Below that point the bailer is faster.

The preceding methods are what may be called preliminary only. They are recommended for outlining the boundary of a deposit and for obtaining preliminary rough samples. None of these methods are recommended for accurate testing in respect to gradation or quality. The only reliable and accurate method of exploring land deposits of sand and gravel is by means of test pits.

Test pits may be dug by hand or by mechanical means. Ordinarily, however, they will be dug by hand. A cross section of 3 by 5 feet makes an economical size in starting at the surface. One man can cut these to a depth of 7 feet. Beyond that another man is required at the surface to shovel material back from the edge of the pit. When the gravel is reached, test pits are usually contracted to a 3 or 4 foot circular cross section. In this operation the man in the pit uses a shovel with the handle cut short to load a small steel or wooden bucket containing from 1/2 to 1 cubic foot of material. His partner at the surface lowers an empty and hoists the loaded bucket by hand or with a windlass. In some deposits such pits can be sunk to 25 or 30 feet without timbering. In others where the gravel is loose, timber may be required at the start. There are several methods of timbering that may be applied to these pits. Several are illustrated in Figures 4, 5, and 6.

Figure 4 shows a common method in which old railroad ties may be used. This type is called cribbing, and there are various ways of applying it which differ only in detail. Cribbing is built upward from the bottom. This requires that the excavated ground will stand without caving for some distance. In starting the timbering, bearer sets are first inserted

to support the crib. These are framed the same as are the short timbers in the crib except that the ends are left from 1½ to 2 feet longer than the width of the crib itself. Holes are dug in the walls of the pit and the bearer timbers inserted. Then the regular crib timbers are built up as shown in the figure. Each may be spiked to the one below.

Mortises and tenons may be cut so that the timbers touch one another, making a tight wall. Material may be saved, however, by leaving a space between the timbers equal to 1/3 the horizontal width of the timber. This space will not allow material to come through into the shaft except in ground under heavy pressure or in material similar to quicksand. As illustrated, 6 by 8 inch ties are used laid on edge. Shoulders are cut 1½ inches deep on each side, leaving a 5-inch tongue. At times notches are cut halfway in each timber with no shoulder on the other side, but this method does not provide as much strength to resist pressure as the framing illustrated.

The pit is sunk as far as it safely can be without caving. The cribbing is then placed and a second lift is excavated, after which a second section of cribbing is placed.

Where the walls of the pit will not stand alone over a distance of 3 or 4 feet, cribbing can be advanced downward with the excavation by using 2 or 3 inch plank. In this method the planks on end and side are alternated in length from inside to outside crib measurement. Each plank is nailed flat to the one above, and the whole hung from bearer timbers on the surface. This provides a tight wall from which little timber can be salvaged. No way is left to inspect or remove samples from the material passed through.

With open cribbing as illustrated, check samples can be taken from any depth at any time, and often the timber may be removed and used over again in another pit.

Figure 5 shows a common form of square-set shaft timbering lagged with plank. This method requires less board feet of lumber and can be advanced as the excavation proceeds. It requires more care in framing. Side shoulders are cut 1 inch deep and tongues are left 3 inches thick. Horizontal members may be 6 by 8 inch ties on edge with 6 by 6 inch posts, although ties may also be used for posts by cutting shoulders longer on one side. The sets are hung from bearer timber on the surface by hanger bolts as shown and a set is added and lagged as excavation proceeds at the bottom.

In ground which will not stand by itself the forepole method shown in Figure 6 is employed. The bearer timbers are placed on the surface and the first timber ring is bolted to them. Plank lagging is cut to reach slightly below the second set and is sharpened to a wedge shape with the bevel on one side only. The lagging is then started with the bevel side of the wedge to the timber and driven down to place with mauls. With lagging driven all around the ring set the ground inside is dug out until posts and the second ring can be set and bolted. A second set of lagging is then driven ahead as before. Very heavy ground may require lagging of double length in order that it may be driven outside the lower ring but inside the next above. Thus the upper end is held in by the ring and will prevent ground pressures from forcing the lower ends too far in to place the new ring of timber.

Lagged sets provide solid walls but single planks may be removed to collect vertical samples.

A more or less recent evolution in constructing the timbered test pit is the use of cylindrical steel casing instead of timber. This casing is either slipped down as a unit and a new section added at the top as the bottom is lowered or the new section is added at the bottom. One operator in Michigan reports a cost for this method of $1.50 per foot with the work done on contract.

Shallow pits designed to penetrate the overburden and extend into the gravel only part way are often dug with portable cranes mounted on caterpillar tractors or with drag-line machines. These machines obviously can not operate in pits of small cross section, hence they can not reach the same depths as timbered pits without digging an excessive amount of

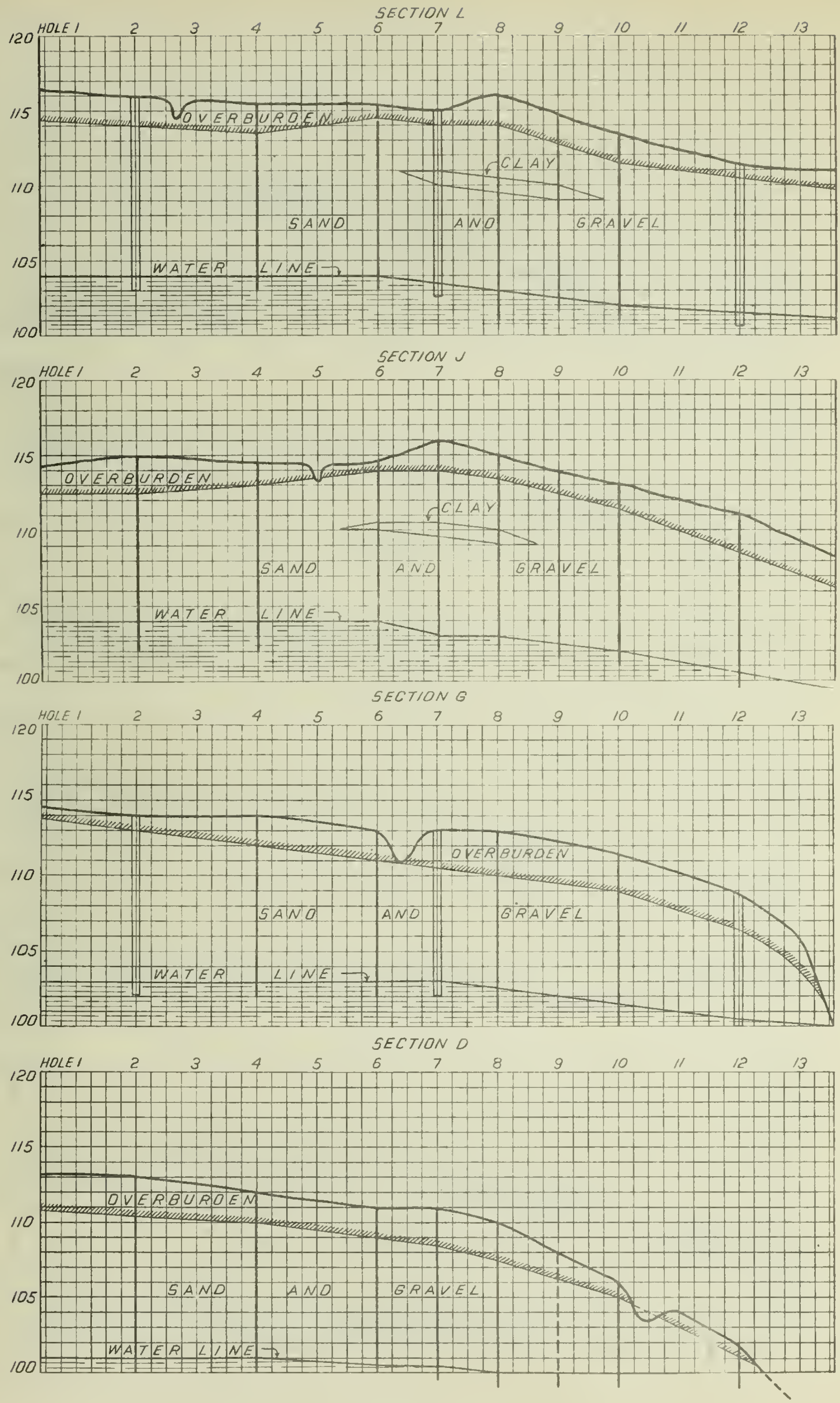

Figure 7.- Sections of deposit, showing character of material encountered by test pits and drill holes

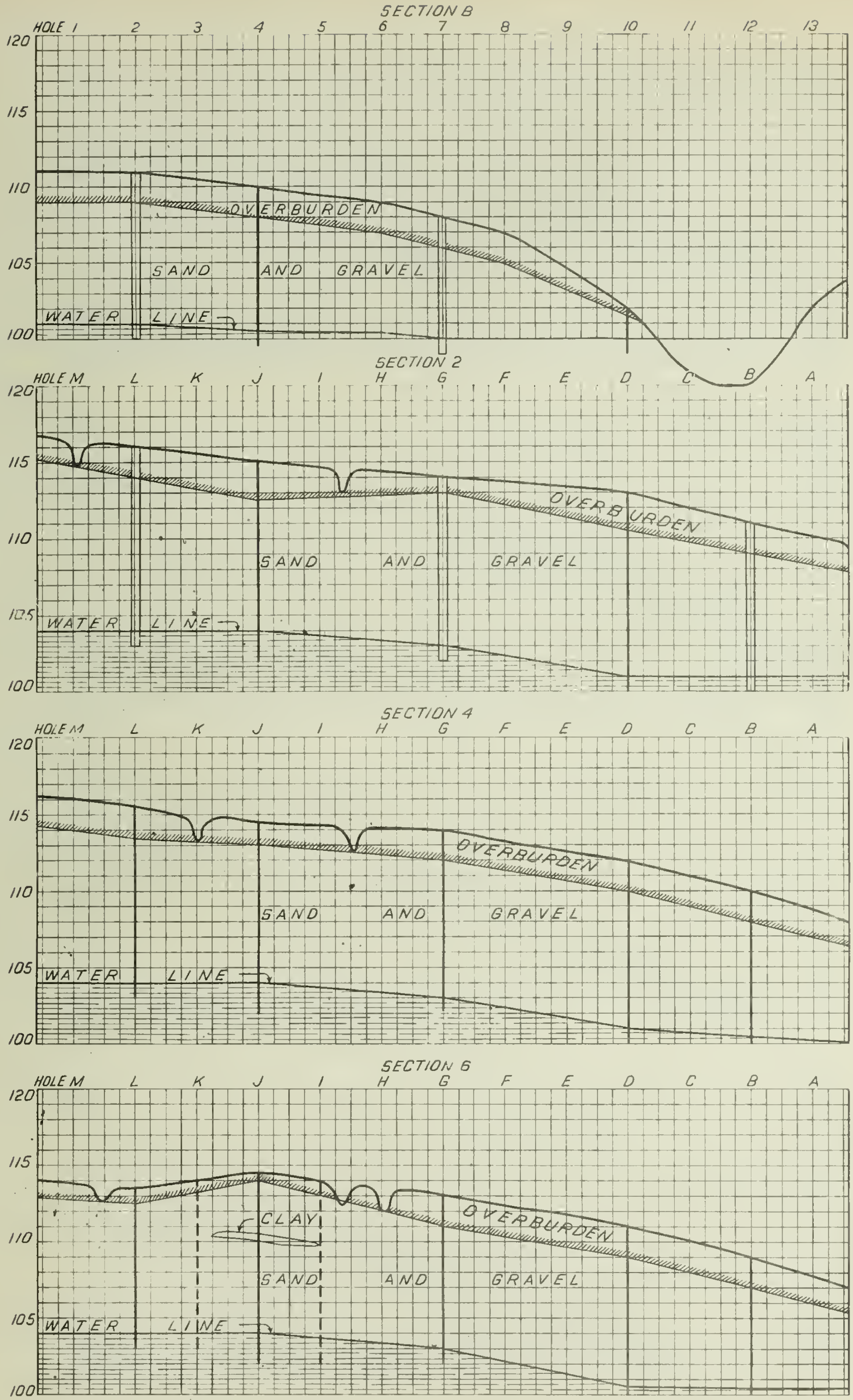

Figure 8.- Sections of deposit, showing character of material encountered by test pits and drill holes

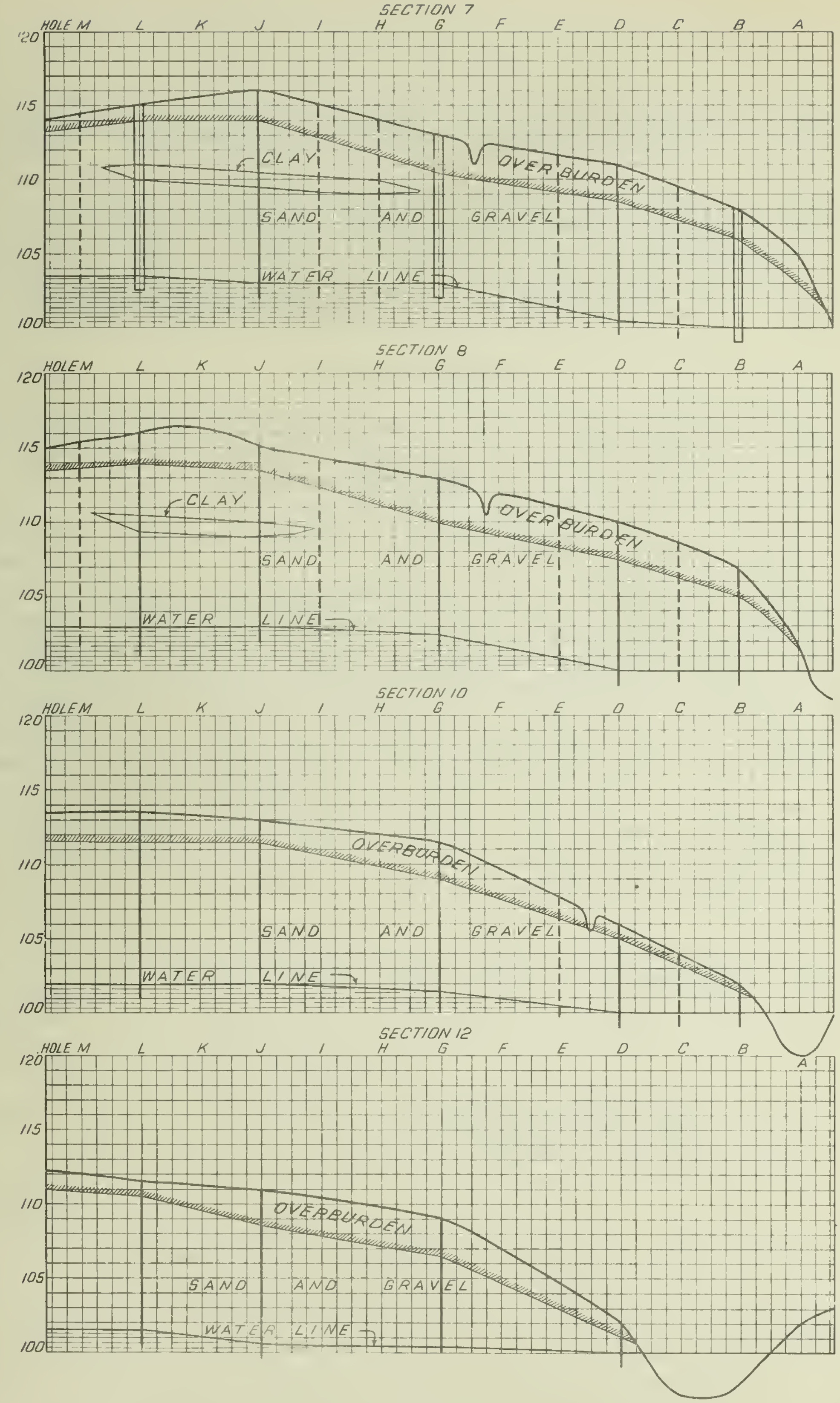

Figure 9.- Sections of deposit, showing character of material encountered by test pits and drill holes

material to provide the necessary slope to the walls to prevent caving. Because of this extra digging they are necessarily more expensive to use in reaching the bottoms of deep or thick deposits.

Test pits afford a means of entering and examining a gravel deposit not offered by any other device. They are more expensive to dig than post holes or churn-drill holes, but their accessability for obtaining accurate information justifies the additional cost.

Ordinarily when water is found, the quantity is too great to permit dewatering the pit by pumping. In this eventuality the test pit is at a disadvantage.

Combination Methods

Various combinations of methods have been employed successfully, as for instance where test pits are sunk to the water level and exploration continues below by churn drill. Another combination is to use the small orange-peel bucket to the water level and follow with the bailer.

Drag lines and cranes have been used to dig shallow pits in the bottom of which timbered pits are sunk by hand.

Another type of combination is predicated upon cost over large areas. In this combination test pits are sunk through the deposit or to the lowest working level. These pits are spaced at rather wide intervals. Between them pipes may be driven or churn drills employed to test the deposit at intermediate points. In this manner erratic changes in the deposit may be spotted and where irregularities appear in the intermediate drill holes they may be replaced by additional test pits to gain a more accurate sample. This latter method is strongly recommended for accurate exploration.

For accurate results the location of test pits or boreholes must follow some systematic plan. If they are spotted indiscriminately over the land surface, considerable difficulty will arise in interpreting and correlating the information obtained from them. If, however, they are located after a regular geometric design, the resulting calculations are facilitated. (See figs. 7, 8, and 9.)

There is much divergence in personal preferences as to the most economical system of spacing test locations. Some prefer to start with post-hole diggers, pipes, or churn drills and when the whole tract is drilled out to check results by digging test pits where drilling has indicated their need. Others start by digging the test pits at rather wide intervals and follow this by check drilling between pits.

Figure 10 represents a 40-acre tract of land with test holes laid out on the corners of 200-foot squares, and with the border holes 60 feet within the boundary.

Figure 11 represents the same tract in which exploration has been started by digging test pits on the corners of 500-foot squares. In this case the border tests are 160 feet from the boundary.

In Figure 10 49 holes are required to complete the first test. If considerable variation occurs in the information obtained from any two adjacent holes as they are sunk, another hole should be put down between them. In this manner, should any hole cut through a clay seam or lens of fine sand not shown in adjacent holes, the extent of this irregularity may be outlined by drilling the eight holes on 100-foot squares left undrilled in the original 200-foot square system. This is illustrated by the "X" marks surrounding hole I-7 in Figure 10. This hole is so designated because it is the point of intersection of an east-west line through I and a north-south line through 7. When a deposit is of such nature that sudden changes of content are found to occur throughout its extent, it may be necessary to spot holes even closer than 100 feet.

So far in following the drill plan of Figure 10 no accurate samples of the deposit have been obtained. The information has indicated the depth of overburden, and the location of irregular variations in the material. The final check will be made by digging test pits at those points where information is needed as deduced from the drill holes.

In Figure 11 the test pits have been put in first. Samples from these pits are carefully examined and tested and where discrepancies occur between the material in two adjacent pits, test holes are drilled between them at 100-foot intervals. In this way the material brought to the surface in drilling may be sufficient when correlated with that from the pits to interpret satisfactorily the meaning of the change between pits. If not, other pits should be dug where one or more holes were drilled.

No hard and fast rule can be laid down as to the arrangement of holes nor the minimum distance that should be left between them. The final result must be arrived at by a study of the cumulative facts as developed by the drilling or test pitting tempered by the knowledge of the engineer or operator.

In order to obtain maximum results from a series of test holes or pits, the data obtained must be recorded in a systematic manner.

As a first step, the operator should obtain a topographic map of the property to be tested. The U. S. Geological Survey has mapped a large part of the more settled area of the country and these topographic maps may be obtained from their office at Washington at nominal cost. Where such topographic maps do not exist, the only recourse of the operator is to the engineer. Property maps without contour lines may be used and holes spotted thereon by the engineer. He can then construct the topographic map from the elevations taken while spotting the holes. As the holes are drilled or the pits dug, the information they afford should be plotted on the map or on notes which are referred to their proper location on the map. With this data complete, the next step is the drafting of vertical sections along lines of tests. These present graphic pictures of what has been discovered and are more easily read than a series of penciled notes.

As an example, Figure 12 has been drawn to represent a topographic map of a 40-acre tract. The border rows of holes have been laid out 60 feet from the boundary as in Figure 10. Intermediate rows of holes are plotted on 100-foot intervals, making 13 rows east and west after the letter designations A to M, and a similar series of rows running north and south corresponding to the numbers 1 to 13. In this manner any hole can be immediately spotted and located by its designation - for example, L-2 or G-7.

The eight test pits were then sunk and results recorded as follows:

Table 5.- Field notes of test pits

Test pit No.	Feet	Elevation	Remarks
L-2			
Surface	0.0	116.0	
Bottom of overburden	2.0	114.0	
Water level	12.0	104.0	
Bottom of pit	13.0	103.0	In gravel
L-7			
Surface	0.0	115.0	
Bottom of overburden	1.0	114.0	
Top of clay seam	4.0	111.0	Blue, sticky
Bottom of clay	5.0	110.0	
Water level	11.5	103.5	
Bottom of pit	12.5	102.5	In gravel

Table 5.- Field notes of test pits - Continued

Test pit No.	Feet	Elevation	Remarks
L-12			
Surface	0.0	111.5	
Bottom of overburden	1.0	110.5	
Water level	10.0	101.5	
Bottom of pit	11.0	100.5	In gravel
G-2			
Surface	0.0	114.0	
Bottom of overburden	1.0	113.0	
Water level	11.0	103.0	
Bottom of pit	12.0	102.0	In gravel
G-7			
Surface	0.0	113.0	
Bottom of overburden	2.5	110.5	
Water level	10.0	103.0	
Bottom of pit	11.0	102.0	In gravel
G-12			
Surface	0.0	109.0	
Bottom of overburden	2.5	106.5	
Water level	8.5	100.5	
Bottom of pit	9.0	100.0	In gravel
B-2			
Surface	0.0	111.0	
Bottom of overburden	2.0	109.0	
Water level	10.0	101.0	
Bottom of pit	11.0	100.0	In gravel
B-7			
Surface	0.0	108.0	
Bottom of overburden	2.0	106.0	
Water level	8.0	100.0	
Bottom of pit	9.0	99.0	In gravel
B-12			
This point in river and no pit sunk. Gravel in river bed at elevation 97.0			

In studying the findings of these test pits it will be noted at once that L-7 cut through a bed of clay 1 foot thick, and none was discovered in the other pits. It therefore became necessary to drill holes around G-7 in order to outline the clay bed. In order to check further it was decided to drill along the north and south lines 6 and 8. Holes L-6, J-6, G-6, D-6, and B-6 and similar holes on line 8 were drilled and their data recorded in the same form as for the test pits above. (See Table 6.)

The drilling of these holes indicated that others would be needed to outline the clay bed effectually. Therefore, holes were drilled as shown by "X" on Figure 12. The original holes in section D discovered an area of fine sand within the gravel, as shown in Table 6, holes D-7, D-8, and D-10. This necessitated drilling of other holes to delineate this sand, also marked "X" in Figure 12.

Table 6.- Field notes of drilled holes

Hole No.	Item	Feet	Elevation	Remarks
L-4	Surface		115.5	
	Bottom of overburden	2.0	113.5	
	Water level	11.5	104.0	
	Bottom of hole	12.5	103.0	Gravel
L-6	Surface		113.5	Creek bed 2 feet below
	Bottom of overburden	1.0	112.5	
	Water level	9.5	104.0	
	Bottom of hole	10.5	103.0	Gravel
L-8	Surface		116.0	
	Bottom of overburden	2.0	114.0	
	Top of clay	5.5	110.5	
	Bottom of clay	6.5	109.5	
	Water level	13.0	103.0	
	Bottom of hole	15.0	101.0	Gravel
L-10	Surface		113.5	
	Bottom of overburden	2.0	111.5	
	Water level	11.5	102.0	
	Bottom of hole	12.5	101.0	Gravel
J-2	Surface		115.0	
	Bottom of overburden	2.5	112.5	
	Water level	11.0	104.0	
	Bottom of hole	13.0	102.0	Gravel
J-4	Surface		114.5	
	Bottom of overburden	1.5	113.0	
	Water level	10.5	104.0	
	Bottom of hole	12.5	102.0	Gravel
J-6	Surface		114.5	Creek 2 feet below
	Bottom of overburden	0.5	114.0	
	Top of clay	4.0	110.5	Blue, sticky
	Bottom of clay	4.5	110.0	
	Water level	10.5	104.0	
	Bottom of hole	12.5	102.0	Gravel

Table 6.- Field notes of drilled holes - Continued

Hole No.	Item	Feet	Elevation	Remarks
J-7	Surface		116.0	
	Bottom of overburden	2.0	114.0	
	Top of clay	5.5	110.5	
	Bottom of clay	6.5	109.5	
	Water level	13.0	103.0	
	Bottom of hole	14.0	102.0	
J-8	Surface		115.0	
	Bottom of overburden	1.5	113.5	
	Top of clay	5.0	110.0	Blue, sticky
	Bottom of clay	6.0	109.0	
	Water level	12.0	103.0	
	Bottom of hole	13.0	102.0	Gravel
J-10	Surface		113.0	
	Bottom of overburden	1.5	111.5	
	Water level	11.0	102.0	
	Bottom of hole	12.0	101.0	Gravel
J-12	Surface		111.0	
	Bottom of overburden	2.5	108.5	
	Water level	10.5	100.5	
	Bottom of hole	11.5	99.5	Gravel
G-4	Surface		114.0	
	Bottom of overburden	2.0	112.0	
	Water level	11.0	103.0	
	Bottom of hole	12.0	102.0	Gravel
G-6	Surface		113.0	
	Bottom of overburden	2.0	111.0	Creek 2 feet below
	Water level	10.0	103.0	
	Bottom of hole	11.0	102.0	
G-8	Surface		113.0	
	Bottom of overburden	3.0	110.0	
	Water level	11.5	102.5	
	Bottom of hole	12.0	101.0	Gravel
G-10	Surface		111.5	
	Bottom of overburden	2.5	109.0	
	Water level	10.0	101.5	
	Bottom of hole	11.5	100.0	Gravel
D-2	Surface		113.0	
	Bottom of overburden	2.5	110.5	
	Water level	12.0	101.0	
	Bottom of hole	13.0	100.0	Gravel

Table 6.- Field notes of drilled holes - Continued

Hole No.	Item	Feet	Elevation	Remarks
D-4	Surface		112.0	
	Bottom of overburden	2.0	110.0	
	Water level	11.0	101.0	
	Bottom of hole	12.0	100.0	Gravel
D-6	Surface		111.0	
	Bottom of overburden	2.0	109.0	
	Water level	10.5	100.5	
	Bottom of hole	11.0	100.0	Gravel
D-7	Surface		111.0	
	Bottom of overburden	2.5	108.5	
	Water level	10.5	100.5	
	Bottom of hole	11.5	99.5	Sand
D-8	Surface		110.0	
	Bottom of overburden	2.5	107.5	
	Water level	10.0	100.0	
	Bottom of hole	11.0	99.0	Sand
D-10	Surface		106.0	
	Bottom of overburden	1.0	105.0	
	Water level	6.0	100.0	
	Bottom of hole	7.0	99.0	Sand
D-12	Surface		102.0	
	Bottom of overburden	1.0	101.0	
	Water level	2.0	100.0	
	Bottom of hole	3.0	99.0	Gravel
B-4	Surface		110.0	
	Bottom of overburden	2.0	108.0	
	Water level	9.5	100.5	
	Bottom of hole	10.0	100.0	Gravel
B-6	Surface		109.0	
	Bottom of overburden	2.0	107.0	
	Water level	8.5	100.5	
	Bottom of hole	9.5	99.5	Gravel
B-8	Surface		107.0	
	Bottom of overburden	2.0	105.0	
	Water level	7.0	100.0	
	Bottom of hole	8.0	99.0	Gravel
B-10	Surface		102.0	
	Bottom of overburden	0.5	101.5	
	Water level	2.0	100.0	
	Bottom of hole	3.0	99.0	Gravel

Having thus discovered two areas of change in character of material within the 40-acre boundary, it was decided to drill other holes to be sure of no further irregularities. Such holes having developed nothing new, the next step was to tabulate the data from the separate holes in groups representing the numbered and lettered lines from which to construct vertical sections.

These notes were tabulated in the same manner as the data from the test pits and are shown in Table 5. From the data in these tables sections were drawn as shown in Figures 7, 8, and 9.

From these sections there is obtained information as to the extent of the deposit except as to its depth. Since all test pits and drilled holes struck permanent water, the depth of the gravel must be determined by churn drill or bailer. Where a number of 40-acre tracts adjoin under the same ownership and the deposit is thick, one deep hole through the gravel on each 40 acres may be sufficient. However, if any two such holes differ considerably in the elevation of the bottom of the gravel others must be drilled to check that irregularity. In thin deposits more frequent depth checks must be made.

The samples taken from the test pits and supplemented by the borings from the holes give information from which an accurate estimate can be made of the gradation and value of the deposit.

In drawing the sections (figs. 7, 8, 9) the horizontal scale was taken as equaling 25 feet for each square and the vertical as 1 foot for each square. Thus, each square represents 25 square feet.

From these vertical sections (five drawn east and west and seven drawn north and south) calculations are made as to the quantities of material available.

Using the east-west sections first, the ground between section L and the north boundary is assumed as having a uniform vertical section equaling section L. Its volume will then equal the square feet in section L multiplied by the distance of that section to the boundary or 160 feet.

In computing the volume of that ground between sections L and J, the areas of the two sections are computed and their mean is multiplied by the distance between them.

By counting squares and multiplying by 25 the cross-sectional area of overburden, clay, or gravel can be obtained. By using mean sectional areas and the distance between them, volumes can be computed in cubic feet, which in turn can be converted to tons or cubic yards as desired.

The five east-west sections and the seven north-south sections can be computed separately as checks.

In making the computation the author assumed the overburden to weigh 90 pounds per cubic foot in place, clay the same, and sand and gravel to weigh 125 pounds.

The results of the computations are as follows in tons:

	East-west sections	North-south sections	Average
Overburden..........	125,918	128,593	127,255
Clay......................	3,978	3,971	3,975
Sand and gravel	805,937	797,961	801,949

Although the preceding method of computation is admittedly not mathematically exact, the author has found it to be sufficiently accurate for practical purposes.

In the exploration of this tract 87.5 feet of test pits were dug and 518 feet of holes drilled.

Assuming a cost of 20 cents per foot for drilling and $1.50 per foot for test pits, the exploration cost would be $234.85, or $0.0003 per ton of sand and gravel above the water level.

The computation for sand and gravel below the water level is a simple multiplication of the horizontal area in square feet by the depth in feet ascertained by churn drilling. This will give the cubic feet of material which multiplied by 125 pounds per cubic foot and divided by 2,000 will give the amount in tons.

This example worked out in elementary detail illustrates the information which can be developed by proper exploration and computation of results.

SAMPLING

Sampling a deposit of sand and gravel is one of the most important functions of exploration. Singularly enough, it is given comparatively little thought by the average operator. All too often a small amount of the dump from each test pit is taken from the most convenient part of the pile and considered as representative of that pile. These portions are thrown together and subjected to careful and exacting screen analyses. The result is then considered as a screen analysis of the whole deposit. The author wishes to particularly emphasize this error, for he has found it to be very common in practice. The actual result of such a test is merely a screen analysis of the material used in the test.

Often operators are surprised and disgusted at the difference in the analyses of material submitted to two or more commercial testing laboratories. The testing laboratory can do nothing more than analyze the material submitted to it. If the samples themselves are unlike, the results from different laboratories are bound to differ. Therefore, in submitting two samples of the same material to different analysts, great care must be taken to assure that both truly represent that material. It is axiomatic that an analysis can be no more accurate than the sample. The care and accuracy of test methods too often far exceeds that used in sampling. It should be quite evident that precision in measurement and test weighing is useless unless equal precision is used in collecting the test sample so that it is truly representative of the material examined.

The following abstract from a definition of sampling is worthy of repetition:

> The correct sampling of a lot of ore (material) is the process of obtaining from it a smaller quantity which contains unchanged percentages of all the constituents of that lot. As there are limits to the accuracy of weighing and determining the constituents _it is necessary only that the error in sampling be smaller than the error in testing_.[16]

The first step in obtaining a truly representative sample of a deposit is the systematic placing of drill holes and test pits over the entire area. Such procedure provides the opportunity for securing accurate samples.

The next step, equal in importance, is the method of taking the sample itself. In sampling test pits the object aimed at is a continuous sample from top to bottom. If the walls of the pit do not require timbering, collection of such a sample is simple. One sampler holds a box against the wall while another with pick or shovel cuts a vertical channel in the wall 6 inches to 1 foot wide, and 2 to 6 inches deep. Collection of material in this manner should be stopped at regular vertical intervals and a new box used. The sample taken should be carefully labeled with the number of the hole and the depths from and to which the

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16 - Peele, Robert, Mining Engineers Handbook: John Wiley & Sons (Inc), New York, 1918, p 1730

material was removed. Vertical intervals vary with different deposits. Where the gravel contains thin sand or clay beds the boxes should be changed at each change in material. Where the deposit is uniform, 5-foot intervals are sufficient. The sampler should keep a record in his notebook as to any unusual occurrence in the deposit such as single bowlders or nests of bowlders, water seams, changes in bedding, etc. His notes should also show from which wall of the pit the sample was taken. It is often advisable to take a similar sample from the opposite wall as a check.

Where the walls of a pit will not stand without timber, the sampling must be done as the pit is sunk. It is seldom that gravel pits will not stand in place for a vertical distance of 2 feet. In other words, it is seldom that the timber must be kept within 2 feet of the bottom. Usually 4 to 5 feet is ample protection. In such cases the wall channel samples are taken just prior to the placing of timber. The procedure is the same except that each 5 feet is sampled separately instead of sampling the whole depth at one time.

In such cases where timber must be kept close to the bottom or in advance of the digging, as in loose dry sand or quick sand, a different method must be employed. Such deposits are by nature more homogeneous in character, and vertical accuracy is not so essential except for notation of the depth at which radical changes occur. In such cases it is often sufficient to take a channel sample across the floor of the pit at each vertical foot in depth.

Sampling by earth augers is of course automatic. The material removed by the auger should be placed in line on the surface. It is then bagged in the predetermined vertical intervals, tagged, and recorded in the sampler's notes.

Sampling of material obtained by bailer or churn drill or wherever water is used or encountered in the hole presents other difficulties in obtaining accuracy. Accurate samples are obtainable by churn drill or bailer only where it is possible to drive the casing ahead of the drill bit or sand pump. Where the bit precedes the casing, contamination is always present from material falling in from the walls of the hole. The presence of water tends to segregate fine sands and clay from coarser material.

These fine materials are maintained in suspension by the churning action of the drill or bailer. Accurate sampling requires that for each vertical interval cut by the drill all material be saved whether coarse or fine. This means that the material in suspension must be saved along with the coarse material. In order to accomplish this, the simplest procedure is to drive the casing 2 feet ahead of the bit or bailer, add the necessary water and drill or bail to the bottom of the casing. All material, including the water, as removed from the hole, should be placed in a box from which no water is allowed to escape before the fines settle out. When completely settled the clear water is poured off and the material in the box bagged and labeled for hole number and depth as before.

When using the small clamshell bucket within casing, the pipe should be driven ahead of the digging a predetermined distance, and the coarse material removed and placed in a continuous pile or ridge. When the bucket has dug to the bottom of the pipe, the water and suspended material should be removed by a bailer and allowed to settle. This fine material is then added to the coarser material and thoroughly mixed before bagging the sample.

When drilling below water level, it is difficult to obtain accurate samples of both fine and coarse material because water enters from the bottom of the casing. This entering water may bring with it fine material from surrounding areas beyond the casing. At best, it provides an excess of water which so dilutes the suspended material that it is almost impossible to obtain the fine and the coarse material from the same vertical section. Many times the incoming water enters so fast that it is impossible to bail the hole dry. This greatly dilutes the fine material and much of it is consequently lost. Therefore, samples taken below the water line must be considered to have been subjected to loss of fines.

When holes encounter bowlders and it is impossible to drive casing, they should be stopped and a new hole started as close to it as possible, or the bowlder blasted. In deposits containing many loose or clustered bowlders, drilling holes may be more expensive than test pitting because of this loss of holes and consequent extra work.

Requirements for extra holes can be determined from the sampler's notes. As an example, holes M-7, M-8, M-9, L-9, K-6, K-9, and J-9 on Figure 12 were drilled to outline the clay seam as shown in dotted lines. Also holes D-7, D-8, and D-10, having shown sand in the bottom, extra holes were drilled at E-7, E-8, E-9, E-10, D-9, and C-7, C-8, C-9, and C-10, to make sure there was no extension of this fine-sand area.

Testing Samples

Comprehensive tests on samples collected by exploration can be made only in a properly equipped laboratory by trained personnel. Few operators are so equipped at their plants. Consequently final testing is usually a matter for the commercial laboratory. However, it is not the function of this paper to describe in detail the exact test methods of the laboratory.

The object of exploration is a rough determination of the size, shape, and value of a deposit. The size and shape have been determined by test pits and holes. The value may be roughly determined by various field tests of the samples. Should these field tests indicate that the deposit has commercial possibilities, then similar samples should be subjected to exhaustive laboratory tests. In only a few instances will it be necessary to have final tests made before a decision can be reached as to the commercial possibilities of the deposit. In such cases a visual inspection of the samples will reveal an appreciable amount of deleterious materials, thus at once placing the deposit on the border line between a paying or losing business. Accurate testing then becomes necessary to determine whether the deleterious matter can be removed profitably and the type of equipment that will be required.

Preliminary or exploration testing will be confined to such items as:

1. The specific gravity.
2. Absorption.
3. The percentage of organic impurities present.
4. The percentage of clay and silt present.
5. The size gradation of sands and gravels.
6. The percentage of soft, friable, and unsound particles present.
7. The presence or absence of deleterious coatings.

The following field tests are suggested for exploration testing, and corresponding laboratory tests as recommended by the American Society for Testing Materials are cited.

Specific Gravity Tests

The true specific gravity of a substance is equal to its weight dry, divided by the weight of an equal volume of distilled water at 4° C. (39.2° F.).

All mineral substances in their natural state contain both open and closed pores. These must be eliminated to determine the true specific gravity. To do this the sample is ground to pass 200-mesh before making the determination.

True specific gravity is seldom used, however, in connection with sand and gravel determinations. The term commonly used is "apparent specific gravity."

The apparent specific gravity of a substance is equal to its weight in air divided by the difference between its weight in air and in water.

Let W_a = the weight of the sample in air.

W_w = the weight of the sample suspended in water.

Then the apparent specific gravity = $\dfrac{W_a}{W_a - W_w}$

The apparent specific gravity may be determined in the field by first carefully weighing a sample of the material in air and then submerging the sample and container in water and weighing again.

Some particles of sand and gravel are much more porous than others. Material submerged in water absorbs water at varying rates, depending on its porosity. Therefore the observed weight submerged must be read instantaneously upon submergence and before absorbtion occurs, otherwise a potential error is introduced. This potential error has led to a modified practice in this test. The sample to be tested is weighed in air after drying at room temperature and the weight recorded as W_{da}. It is then placed in water and allowed to stand 24 hours, or until saturated. The sample is then surface dried and weighed and this weight recorded as W_{sa}. It is then submerged in water, weighed and the weight recorded as W_{sw}. The apparent specific gravity of the saturated sample is then

$$\frac{W_{da}}{W_{sa} - W_{sw}}$$

Absorption

For a more accurate laboratory method see A.S.T.M. designation C-68 and D-55 for sand and C-86 and D-30 for gravel.

The absorption test is performed in determining the specific gravity. $W_{sa} - W_{da}$ equals the absorption.

Organic Impurities

Organic impurities, such as leaves, twigs, peat, and coal, can be eliminated by burning. As low as 0.05 per cent of certain organic impurities if contained in aggregates are injurious to concrete.[17] On the other hand, appreciably greater amounts of other organic impurities such as lignite are permissible.[18]

A common test for organic impurities consisted in recording the weight of the sample in air when dried at less than 110° F. to constant weight. The sample is then heated to 400° to 500° F. for about one half hour, cooled, and again weighed. The difference in weight was recorded as the loss in organic matter.

However, small amounts of impurities would present such small losses by this method as to require precision balances to detect them. Since these balances are not available in the field the colorimeter test should be made as follows.[19]

Fill a 12-ounce graduated prescription bottle to the 4½-ounce mark with the sand sample. Add a 3 per cent solution of sodium hydroxide to the 7-ounce mark. Shake thoroughly and let

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17 - Abrams, Duff A., Effect of Tannic Acid on the Strength of Concrete: Struct. Mat. Research Lab., Lewis Inst., Chicago, Bull. 7, 1922, p. 27.

18 - American Society for Testing Materials, Effect of Coal and Lignite in Sand for Concrete: Proceedings, 1929, pt. 1, p. 328.

19 - American Society for Testing Materials, Standards, pt. 2, 1930, p. 154.

stand 24 hours. Observe the color of the clear liquid above the sand. Clear or light yellowish color indicates sand satisfactory for concrete so far as organic impurities are concerned. Straw color indicates sand suitable for unimportant concrete. Dark colors indicate sand unfit for concrete.

Sand showing straw or dark color should be examined for lignite. If present, the sand should not be condemned on account of the colorimeter test until it is confirmed by mortar strength tests.

The more accurate method for determining the percentage of organic matter present is described under A.S.T.M. designation C-40.

Clay and Silt

In determining the amount of clay and silt present, record the weight of the sample as dried in air to constant weight, as W_d. Place the sample in a suitable pan in which it can be covered with water. Stir thoroughly and pour off the water, being careful that no sand is poured off. Add more water, stir, and pour off. Repeat until the water removed is clear. Dry the remaining sample to constant weight and weigh. Record this weight as W_c.

$W_d - W_c$ = weight of clay and silt removed.

$$\frac{W_d - W_a}{W_d} \times 100 = \text{per cent of clay and silt.}$$

A similar method is followed by some operators and has the advantage of using larger samples, thus avoiding possible errors in sample reduction.

A small revolving screen (3/8-inch mesh) is mounted in a water-tight wooden or metal box. The box is partly filled with water so that the material in the screen is submerged. The sample is then placed in the screen and revolved. The material retained in the screen will be gravel and that passing through to the box will be sand, clay, and silt. The latter is allowed to settle and the water drained off. The sand, clay, and silt are then subjected to the clay-silt test as previously described.

This test has the advantage of showing the character of the clay and its effect in adhering to gravel, collecting fines, accumulating as balls, etc., under conditions simulating plant practice.

The A.S.T.M. tests for determination of clay and silt in fine and coarse aggregates is termed the "decantation test" and may be found under the designation D-136 for sand and D-72 for gravel.

Size Gradation

The field sample is now ready for a sieve analysis. The first step in the field test is a separation of the sand from the gravel.

The dried sample from the clay and silt determination is weighed and the weight recorded as A.

The sample is then placed on a screen perforated with 3/8-inch round holes and agitated to refusal - i.e., till no material will pass.

The material passing the screen is weighed and the weight recorded as S. The material retained is weighed and recorded as G.

Then S + G = A

and S/A x 100 = per cent of sand.

G/A x 100 = per cent of gravel.

Sample S is then sieved through the testing sieves for sizing. Various sets of sieve standards have been and are being used. For the purpose of uniformity it is recommended that the A.S.T.M. sieve standards as given under designations E-11 and E-17 be used.

When so used the sand sample of weight S will be sieved over and through sieves Nos. 4, 8, 16, 30, 50, and 100. The weight of the material passing each will be recorded in tabular form as follows:

Passing	Retained on	Weight
3/8	4	t
4	8	u
8	16	v
16	30	w
30	50	x
50	100	y
100		z
	Total = S =	(S)

Then $\frac{t}{A} \times 100$ = per cent passing 3/8 inch and retained on No. 4 sieve, etc.

Sample G will then be passed through screens having the following sizes of round perforations: 3 inch, 2 inch, 1-1/2 inch, 1 inch, 3/4 inch, 1/2 inch. The weight of the material passing each and retained on the next smaller opening will be recorded in tabular form as follows:

Passing, inch	Retained on, inch	Weight
3	2	o
2	1-1/2	p
1-1/2	1	q
1	3/4	r
3/4	1/2	s
1/2	1/4	t
	Total = G =	G

Then $\frac{o}{A} \times 100$ = per cent passing 3-inch and retained on 2-inch, etc.

Local conditions may require the use of sieves and screens with openings of other than the sizes shown, but the procedure for testing will be the same in any case.

For A.S.T.M. standard methods see designations C-41, D-7, and D-19.

The major questions which arise in the mind of the operator in valuing a sand and gravel deposit after it has been demonstrated that sufficient volume is present are -

1. The percentage of sand present.
2. The percentage of fines that must be removed from the sand to make it marketable.
3. The percentage of fine gravel that must be wasted.
4. The percentage of gravel present.
5. The percentage of oversize that will require crushing.

With properly collected samples the preceding sieve tests will provide the proper answers to these questions.

Soft, Friable, and Unsound Particles

This test applies to the gravel only and is made by what are known as the soundness and abrasion tests. Both of these tests require laboratory apparatus for accuracy. However, the experienced prospector will be able to appraise closely the percentage of soft, friable, and unsound particles present by a visual inspection, for which a portion of the sample is spread on a table and the deleterious particles separated and counted. This count compared to the total number of particles gives the percentage.

The sodium sulphate test is commonly used for a soundness test of material showing no soft particles on visual inspection. This test can be made in the field as follows.

A saturated solution of sodium sulphate is prepared. It is important that this solution be saturated and kept in this condition. The gravel sample is then placed in this solution and allowed to stand for 18 to 24 hours. It is then removed, oven dried, and again immersed in the saturated solution for another 18 to 24 hours. This alternate soaking and drying is repeated five times. The test should be carried on in a warm room. This field test will expose unsound material and because of the range of temperature changes will be somewhat more severe than the laboratory test.

The A.S.T.M. designation for laboratory soundness test is C88-31-T and C89-31-T.

Any preliminary test of new gravel deposits should include a laboratory test.

Deleterious Coatings

Gravel particles are often coated with deleterious siliceous, calcareous, or argillaceous materials which if not removed cause unsound concrete. There is no particular test for these coatings other than visual inspection of the washed sample. The apparatus mentioned under the clay and silt test, page 36, will be valuable in facilitating inspection of the gravel under conditions simulating actual plant washing practice.

In addition to the preceding tests the material should be tested for abrasion loss. There is no practical field test for this and it must be performed in a laboratory having the proper equipment.

The A.S.T.M. designation for abrasion test is D289-28T.

Splitting Samples

Often it is desired to split the sample to have it tested by independent laboratories. In doing this exceeding care must be taken, or erratic results will follow. Here is where many operators have difficulty.

Sand or fine gravel samples can be split without much danger of descrepancy, but as the material increases in size the difficulty of accurate cutting increases proportionately.

A method commonly used is the cone and quarter method. This is employed extensively in metallurgical plants and is more or less standardized. Sand and gravel operators have adopted it in many cases without inquiring as to whether it was adapted to their material or not. As a matter of fact, it is not. The cone and quarter method is applicable only to material that has been previously screened to approximately the same size. With a mixture of sand and gravel, conditions are changed and the method is not accurate.

In the cone and quarter method, the sample is spread out on a metal or wooden floor in a rough circular pile a few inches thick. The sampler then takes a small shovel and throws material from the periphery of the pile to the center, letting each shovelful fall on the cone which is thus built up. After thorough mixing in this manner, the operator divides the cone into four quarters either with his shovel or by inserting a plate vertically through

the pile in one direction and separating the two halves. The plate is then turned at right angles and the two piles separated into four, or quarters. Each quarter then is supposed to represent accurately the whole original sample. This works well with sized material but where coarse and fine particles are indiscriminately mixed in a sample, the coarser lumps roll to the outer edge of the cone and the fine remain in the center. This size segregation prevents accuracy in quartering for it is impossible to assure equal rolling in all directions from the top of the cone.

Better results are obtained in splitting samples of mixed sizes by the use of automatic samplers. These are boxes in which partitions are placed whereby material falling within one section will be diverted to one direction while material in the next space will pass to another direction. In their use the sample is shoveled onto the sampler which automatically separates it into two representative portions.

Figure 13 illustrates a simple device for this purpose. Obviously, the width between partitions must be greater than the largest particles in the sample.

The deposit having been discovered by the prospector and properly examined by exploration and testing, the work of the prospector ceases. It now devolves upon the operator to evaluate the results obtained from the prospector's tests with the market requirements, to ascertain whether the deposit has an economic value. If there is an excess of fine sand, or fine gravel, he must ascertain how great this excess will be from his screen analysis of the samples. This will show him how much material must be wasted or what markets he must investigate for its disposal. From the same analyses he can determine the size of his treatment units for a given capacity requirement, and thus construct a flow sheet for the proposed plant. Thus prospecting and exploration are, or should be, the foundation upon which the operator bases the value of his deposit as well as the location and design of his plant.

APPLICATION OF PROSPECTING AND EXPLORATION METHODS TO VARIOUS TYPES OF DEPOSITS

Residual

This type of deposit results from the weathering of bedrock in place. No transportation has taken place and the sand and gravel is usually intimately intermixed with clay and bowlders.

Prospecting methods, in addition to a study of local geology, will involve an examination of the physical character of fragments exposed on the surface. Search will be limited to the tops of well-rounded ridges having thick, heavy subsoil and lying between well-defined drainage basins. Favorable areas are those portions of the country underlain by limestone or other decomposable rock and which have not been overrun by glaciers.

Since residual deposits are seldom over a few feet thick, exploration will consist of shallow test pits and boreholes sunk either by auger or churn drill. Many holes will be lost because of encountering bowlders, and particular care must be exercised in determining the percentage and character of fine sand and clay present. Careful examination and testing of the commercial sand and gravel will be necessary because of the presence of "rotten" or badly weathered stone which will not stand up under hardness and abrasion tests. Because of the usual high percentage of flint or chert particles, care must be given to see that they pass engineer's specifications for concrete aggregate. Residual deposits are not as a rule favorable for commercial development, and when exploitation of them is contemplated unusual care is needed both to ascertain their physical composition and to design the proper type of plant that will eliminate the impurities and assure a market material that will pass specifications.

Fluvial Deposits

Fluvial deposits vary considerably in type and will require different methods both of prospecting and exploration. Land deposits may be classified into:

Alluvial cones and plains,
River terraces and flood plains, and
Deltas.

The method of formation of these several types has been discussed previously in this report.

Alluvial cones are recognizable from their proximity to mountain ranges and the character and distribution of the particles. Alluvial sands are usually not well rounded, but fairly angular. They will contain few pieces of decomposable minerals such as limestone, feldspar, etc. Gravels, on the other hand, will be subangular to well rounded with smooth surfaces, and if transported only short distances may contain small to large percentages of decomposable minerals. Deposits will contain comparatively small amounts of fine sand and clay, but the coarser sand, gravel, and bowlders may be more or less segregated in irregular lenses. Clusters of bowlders may also be present near the base of the cone. Alluvial cones will range from a few feet to several hundred feet in vertical thickness.

Prospecting for this type of deposit will involve principally a study of local geology and topography.

Exploration will involve test pits and churn or other types of drill holes. Augers will be useless except in determining the depth of overburden.

In large cones approaching alluvial plains in size, the search will be directed to ascertaining that portion of the deposit which contains the best commercial gradation. Nests of bowlders are not objectionable, but they will require crushing machinery. Test pits supplemented by churn drilling will probably be the best methods to use. It will seldom be necessary to ascertain the depth of large cones because of their great thickness and extent.

Ordinarily there will be less need for determining the percentage of fine sand and clay because of their limited presence due to the manner of formation. When present, they will be in such form as will require a comparatively minor treatment for elimination. Soft or rotten stone will not usually be present. Careful testing of pit and drill-hole samples will be necessary, however, to ascertain the percentage of gradation present, in order that the pit and plant may be located on the most favorable portion of the deposit. Careful testing may develop areas wherein sand may be produced separately from gravel. Usually such deposits will be well drained and there will be no water remaining in drill holes. Depth drilling will be necessary only to determine the commercial quantity available.

Alluvial plains are similar to cones where they abut steep mountain ranges, and prospecting and exploration methods are the same as for cones. Where they extend great distances from the mountains, as in the great plains area, the deposits will not attain such great thickness and the ingredients will be better graded with fewer bowlders and more fine sands. There will be more clay intermixed with the material also, as the distance from the mountains increases until finally alluvial plains merge into flood plains.

In prospecting, a knowledge of the local geology will be of considerable help in locating ancient river courses. In extended plains, local topography will not help greatly, but vegetation may indicate deposits. The prospector will do well to be guided by exposures in river banks, railway, and highway cuts and in well records.

Exploration will be best accomplished by test pits supplemented by drilled holes. The churn drill will be useful because some deposits will be found to extend below the subsurface

water level. Augers may be used in determining overburden depth and occasionally in obtaining sand or gravel samples. The casing and bailer method may also be found to be advantageous.

Care must be exercised in testing for clay and fine sand content as well as the percentage of fine gravel. All gravel must be carefully tested for hardness and abrasion. The same scrutiny must be made for gradation also. Since this type of deposit varies greatly in thickness, occasional holes or pits must be sunk completely through the deposit to be sure of its depth. The subsurface water level must be accurately determined, since the entire method of exploitation may depend upon the extent of the deposit below water.

River Terraces

Old rivers left considerable quantities of sand and gravel in their beds and banks. As continental elevations took place, or the river cut deeper in its bed, these deposits are left as terraces along the valley walls parallel to the stream. In such areas topography will be the principal guide for the prospector. Exposures in the river banks and along tributaries indicate deposits above, where terraced topography is present.

Observation should be made for the usual physical characteristics of the particles of fluvial deposits namely, angular sands and subangular to rounded smooth-faced gravels.

Exploration will employ test pits and the several types of drilling. Deposits will not usually reach great thickness and therefore their depth must be ascertained. Size segregation may be prevalent, hence exploration will seek to determine the gradation of the material. Abrupt changes in gradation may occur, therefore drill holes or pits may show erratic results over short distances. Many extra holes will be needed to interpret these discrepancies. As such places are usually well drained, permanent water will seldom be found in holes in terrace deposits. Occasional silt or clay beds will require careful testing as to the percentage of such material or the outlining of such beds by drilling so that they may be avoided in later exploitation. Auger holes may be used to advantage in testing overburden and outlining area and thickness of surface sand or clay.

Flood Plains

Flood-plain deposits are formed by the material carried over the river banks during periods of high water. Each single deposit is necessarily thin but where a river periodically overflows the same area these may be built up to considerable thickness.

Regional topography will be the principal guide to the prospector in his search for this type of deposit. His search for commercial material will be materially assisted by records of wells and cellar excavations as well as by exposures in railway and highway cuts, dry gulches, and banks of tributary streams. They are usually covered by soil of varying depths and the deposits themselves are likely to be badly intermixed with beds of clay or silt. Gradation is generally erratic and particular care must be taken to determine the percentages of clay, silt, fine sand, and commercial sand and gravel present. Usually there will be a much higher percentage of sand and fine materials than of coarse gravel in flood-plain deposits. The bars in rivers traversing flood plains will be found to be badly contaminated with silt also.

Test pits and all types of drilling are the methods employed for exploration. Holes must be spaced at shorter intervals because of the sudden and erratic changes in the ingredients. Ordinarily flood-plain deposits are above subsurface water levels, but in some cases they are not. Therefore, it is important to determine the permanent water level. Great care must be used in ascertaining not only the percentages of fine and coarse materials present,

but the gradation of commercial sizes. Since the deposits are usually thin, their depth must be investigated. Flood-plain deposits rarely contain clean sand nor do they contain sands in commercial gradations. This means that equipment must be installed to eliminate the undesirable sizes; hence, it is important to know beforehand the percentage proportion they represent.

Commercial sand and gravel in such deposits, however, will usually be quite free from soft particles in the form of rotten stone or decomposable material. The problem to be solved by exploration is one of extent and gradation in order that a commercial material may be made from a noncommercial natural deposit.

Delta Deposits

Deltas are formed in the ocean or lakes at the mouths of rivers. Usually before reaching the sea or lake the river current has so lost its velocity as to render it incapable of transporting material coarser than fine sand. Therefore, delta deposits are usually deficient in commercial gravel. However, during flood periods the velocity may be so increased as to allow it to bring down and deposit coarse material. Also, where the debouching river meets strong wave or tidal action the fine material may be caused to remain in suspension by agitation until carried far beyond the mouth, and only the coarser product be deposited in the delta. Thus considerable sorting is possible.

Prospecting on deltas is simple in that it is confined to relatively small areas bounded by the extreme migration of the river and the sea or lake. Old abandoned river courses may present outcrops in their banks. Any artificial excavation will present material for study also.

Exploration will be by means of auger or drill hole, since the proximity of the permanent water level to the surface prevents the use of test pits.

Because of the multiplicity of materials deposited and the changing course of the river, extreme care must be used in exploration to determine both the kind of material, its size gradation, percentage composition, the presence of soft or inferior particles, and the presence of debris.

River Bars

Underwater fluvial deposits are found within the banks of rivers or streams. They form shifting bars built by the water currents. The river may cut away a bank at one point and deposit the material in a bar at another, which in turn may be cut at a later date and deposited still lower in the course of the stream. Under favorable conditions a river may break through its bank and abandon the old stream bed entirely.

Nearly all river-bar deposits are visible at one season or another. In some rivers irrigation or navigation dams have raised the water level and covered previously exposed bars but where such artificial barriers have not been erected shifting currents will usually expose deposits at some time. This makes prospecting fairly simple. Since running water sorts and grades materials, coarse sands and gravels will be expected in the swifter currents and as the velocity decreases the size of particles also decreases.

Any agency which causes a deflection of the stream course or decrease in velocity of its current also causes deposition. Thus sand and gravel deposits may be expected in bars on the inner sides of bends. A tributary entering a river from one side tends to divert or retard its current, and deposits may be expected on the up-river side of the mouths of tributaries. Shoal places in the river below deep channels tend to retard the current and material is deposited above them. The prospector must depend upon his knowledge of, or information he can

obtain concerning, the habits of the river currents, both as to present and past history, for likely sites in locating river bars. Where the gravel is not exposed to view the prospector may use a long steel bar and sound for gravel in the river bed. Usually, however, prospecting for river-bar gravels is done by dredging. The dredge is spotte.. at a likely site and digs from the bed until it can be determined whether the material is commercial or not.

Test pits in river bars are useless as they can not be extended below water level. Bar gravels, however, are ordinarily very loose and can at times be recovered by augers or drills within casing. The small orange-peel bucket operating within a casing should be helpful. as should also the bailer and casing.

Either of these methods promise success when they can be located on the bar where it outcrops above water level. For underwater work, however, they are not so successful because of the difficulty of holding casing steady from a boat on the surface. This does not mean that it can not be done, but the cost of such equipment together with the difficulties encountered increase the exploration expense until most operators prefer first to sound by bar and then spot the dredge.

Exploration of bars should develop the presence or absence of river debris such as sunken logs, tree trunks, waste materials from up-river sources, and strata of silt or fine sand. In some cases an abundance of shells may impoverish an otherwise commercial deposit or so alter conditions as to require special machinery for their elimination in the treatment plant

Since any exploration must be below water level, experience is necessary in collecting samples because of the danger of misinterpretation due to the partial removal of silt and fine sand by the water while in the act of collecting the sample. Care must be taken to see that there is not too much clay or silt present. As an example, one river deposit is reported as having several feet of good material in two beds between which lies a bed of heavy blue clay. Dredging the clay with the gravel so mixes the material that it is impossible to obtain a clean sand or gravel. Other types of clay cause little trouble and are easily washed out. Therefore, if clay is encountered, it should be carefully investigated as to its character and probable effect on the finished product.

There need be less care taken in determining the gradation of the material from river bars because in the recovery by dredging excess quantities of certain sizes may usually be wasted overboard without additional disposal cost. There is a need for information as to the approximate gradation, however, in order that the proper equipment may be installed for eliminating the waste sizes.

Care should be exercised in determining the amount of foreign industrial waste occurring in the deposit. As an example, in some river deposits considerable quantities of coal are encountered, resulting from refuse from up-stream coal-mine waste. This coal must be removed from the sand and gravel, and when it is present in sufficient amount may present a serious problem calling for special equipment. Also the amount of sunken debris present may determine the type of dredge necessary. Exploration in river beds which have been previously dug over for their gravel content encounters another type of difficulty. Dug-over river bottoms are often left in a series of inverted cone-shaped depressions. These provide excellent catch basins for all sorts of refuse brought down the river. They may also be found to collect bowlders in nests from the sides of the cones or rolled along the bed of the river in flood periods.

River beds and bars are seldom over 50 feet thick; hence, it is essential that the thickness be determined both in order to calculate the tonnage available and to indicate the type of equipment necessary.

Marine and Lake Deposits

In searching for sand and gravel deposits in the coastal plain areas the prospector will need a knowledge of geology as well as topography. He will recognize deposits of marine origin both by the shape and position of the deposit as well as the physical characteristics of the particles.

Marine gravels found on the tops of well-rounded hills or ridges will differ from residual deposits in like topography in that they will be well graded and the particles usually well rounded. Few angular sands will be found and none of soft materials. Gravels will be well rounded with smooth faces having a dull polish, while residual gravels are rough and usually have pitted faces.

Likely sources will be terraces parallel to the seashore and flat-topped elevations between well-established drainage areas within coastal plains. Scrutiny should be given to stream banks cut through low hills or terraces, and railway and highway cuts. Vegetation may be a helpful guide also. The prospector operating in coastal-plain areas may encounter terraces parallel to seashores having materials which do not show marine characteristics. This is true along the Atlantic coast where some of the terraces are alluvial plains which have been washed and eroded from old shore lines. He should also not overlook river bars in coastal plains.

Regional topography and geology will be his principal guides. Observations should include inspection of exposures in stream banks cutting through terraces, as well as artificial excavations of all sorts.

Exploration will employ test pits and all classes of drill holes. The objective sought will be particularly to locate such portions of the terraces or hilltops as contain as near the proper-sized gradations as possible. Marine deposition tends to group gravels in fairly well-sorted sizes; therefore, fine sands may be found in one place and coarser material or bowlders in another near-by. Often the commercial materials will be found overlaid by heavy silt or clay, the thickness and character of which must be carefully determined for its effect on later operation. There will be little difficulty with permanent water in the higher terraces, although this becomes a problem in more recent uplifts that are raised only slightly above sea level. Little trouble will be experienced with soft particles because of the long-continued abrasion both during transportation to its undersea location and later by wave action on beaches to which the material has been subjected. However, some marine gravels do contain limestone particles which have been formed in the sea by chemical means or as the result of organic remains. These usually will have been so worn by wave action as to lose their soft portions.

Of equal importance in marine deposition are the large offshore deposits along the continental shelf. These are similar to terrace deposits except that they have not been raised above sea level. They are formed by the deposition of material brought down by rivers and streams and are therefore of fluvial origin. Suspension in the heavy sea water is more potent than in fresh water and this combined with agitation from wave, tidal, and current action tends to carry finer silts and clays beyond the continental shelf into deep-sea levels. The sands and gravels are thus deposited close to shore on the continental shelf and form important sources of commercial material.

Prospecting for such deposits consists of examination of the adjacent beaches and the formation of offshore islands. Any approved method of making undersea soundings may be employed also. Probably the most often-used method is the actual digging of the material with dredges as is done in river-bar deposits.

Exploration is by dredge alone. The principal information sought is the depth below water surface and thickness and character of the material. This combination determines the

type of equipment necessary for exploitation. The habit of local tides is also a contributing factor in this problem, for material accessible at low tide may be out of reach if the tide variation is great. Harbor protection is also important, as dredging equipment is not ordinarily designed to withstand stormy seas.

Size gradation is important because of the tendency of wave action to sort material in size groups. Abundance of shell or coral formation may complicate production of commercial material. Therefore the percentage of these materials should be carefully ascertained by adequate exploration before production on a large scale is started.

Similar methods are employed in prospecting for gravels along the boundaries of inland lakes.

Occasional remnants of gravel deposits of marine origin are found far inland marking the shore lines of ancient seas. These are commercially unimportant as a class, however, inasmuch as more recent erosion has so reworked them that they are at present classed as fluvial. Deposits along the old boundary of the great Salt Lake are exceptions. These resemble alluvial fans or narrow plains in that they abut steep mountain ranges. Exposures in recent stream beds form the principal methods of examination for the prospector. Exploration methods are similar to those used in coastal plain terraces.

Glacial Deposits

It is essential that the prospector working in areas within the boundary of glacial action have a knowledge of the geologic history of the region. With such knowledge his understanding and interpretation of the various forms of deposit encountered will be greatly facilitated. This involves a knowledge of the origin of the formations and the resulting topography.

In his search the prospector will observe the position and appearance of isolated hills and ridges, the abundance of small lakes, the presence or absence of sharp topographic relief and the markings left on the surface of outcrops of bedrock. Grooves and striations on outcrops show the direction of ice movement. The relation of the long axis of ridges or the alignment of isolated low hills will indicate whether the deposits are moraines or eskers. This in turn will indicate to him the type of material he may expect below the land surface. Examination of particles where exposed in natural or artificial cuts will indicate glacial source by the striated or grooved surfaces of the gravels.

Sands containing particles of feldspar and other decomposable minerals indicate transportation over comparatively short distances or possible source in glacial deposits.

Sounding bars may be used to advantage through thin overburden to detect gravel below, although the prospector's principal reliance will be upon the topography and his knowledge of the geology of the region supplemented by examination of natural or artificial cuts and excavations.

Exploration will be by test pits, drill holes, and soundings. The latter, while of principal interest in prospecting, may be useful in outlining deposits under thin overburden. For the same purpose, auger holes are employed, although much difficulty may be expected from loss of holes through encountering bowlders. The same difficulty will arise in the use of casing with drilled holes. Therefore, accurate sampling will necessitate the digging of frequent test pits.

Especial care must be taken in examining glacial deposits to determine accurately the proportion of fine sand and clay. Moraines especially are heterogenous deposits of unsized materials which necessitate careful systematic examination because in the absence of bedding or segregation of any sort no assumptions can be made from superficial or cursory exploration. Such deposits must be tested over their whole area and throughout their full depth. This means that vertical examination is of equal importance with lateral examination.

The possibility of the material containing decomposable minerals which would be harmful in concrete construction must also be carefully ascertained. When discovered, their character and percentage must be determined as accurately as possible. This type of impurity may be found both as fine sand and coarse gravel or even bowlders.

The gradation is important, not only for its relation to market requirements but to determine the proportion of bowlders present that may require a crushing plant. This is not easy in this type of deposit because of the heterogenous mixture. Even with elaborate systematic exploration, considerable latitude in interpretation of results will be necessary. This must be tempered by the judgement and experience of the prospector.

Fluvioglacial Deposits

Fluvioglacial deposits resemble fluvial deposits in their formation. They differ in that they are found in close proximity to glacial deposits or the boundary of glaciated areas. Material contained in them will show an absence of clay and fine sand and considerable grading will be apparent. Sands will be angular but gravels will be rounded and smooth-faced. Less quantities of decomposable material will be present.

Outwash cones or plains will resemble alluvial cones and plains except that they will not abut mountain ranges. They will, however, emerge from old glacial moraines. Lateral and ground moraines will have been reworked and the material deposited in river bars or terraces along the course of the glacial streams or carried on to form outwash plains. Therefore, a knowledge of local and regional geology and topography is helpful.

Prospecting methods will be of the same type as for similar fluvial formations.

Fluvioglacial deposits will contain material showing the characteristic striations and grooves of glacial abrasion.

Exploration should develop the size and depth of deposits, the gradation of the materials, and the percentage of fines. The presence and character of decomposable material should also be determined. There will be ordinarily little clay present and materials will be fairly well graded even to size segregations to some extent. Vertical depth may range from a few feet to over a hundred feet; hence, depth must be determined.

TYPICAL EXAMPLES WITH COSTS

The United States Bureau of Mines has published a number of Information Circulars on sand and gravel mining methods and costs, of which the following table lists a few.

Typical examples of sand and gravel operations

Publication No.	Company	Location	Type of deposit
I. C. 6420	Menantico Sand & Gravel Co.	New Jersey	Marine
I. C. 6421	Ohio River Sand Co.	Kentucky	River bar
I. C. 6537	East Texas Gravel Co.	Texas	Flood plain
I. C. 6580	Ward Sand & Gravel Co.	Michigan	(Fluvioglacial
			(Moraine
I. C. 6581	Dallas Washed & Screened Gravel Co.	Texas	Fluvial
I. C. 6582	J. K. Davison & Bro.	Pennsylvania	River bar
I. C. 6592	Seaboard Sand & Gravel Co.	New York	Moraine
I. C. 6607	Consolidated Rock Products Co.	California	Alluvian fan

I. C. 6420

The deposits on the property of the Menantico Sand & Gravel Co. consist of two types. The older or Highland deposit is probably typically marine with little or no overburden and supporting a weak growth of scrub oak and pine. The Lowland deposits are found along stream courses and represent marine materials reworked and left as fluvial deposits.

Exploration was carried on by means of test pits limited in depth by the permanent water level. Below this depth hand-drilled test holes were put down on the corners of 100-foot squares at a contract price of $1.50 per foot. Because of insufficient information obtained by this method, the clamshell bucket working within a 12-inch casing was adopted. This is now used to the water level. Below that a 4-inch pipe is sunk and a 3-inch pipe bailer is operated within it to remove the sample. The lower end of the pipe is filed to a rough cutting edge, and a flap valve is placed about 3 inches above. By churning the bailer up and down by hand the material within the 4-inch casing is loosened and brought to the surface.

Three men are used to operate the bailer. They average about 50 feet of hole per day at a cost of 30 cents per foot of hole.

I. C. 6421

The Ohio River Sand Co. operates wholly in river-bar deposits in the Ohio River. The company uses churn drills to prospect bars that outcrop above water level. It owns two islands in the river that have been prospected in this manner. For underwater bars a sounding rod is used in the hands of the operator from a small boat. By this means, gravel beds are "felt" or "sounded" beneath the river bed of fine silt. When gravel is located by sounding, the ladder dredge is moved to the spot and exploration continued by actual digging.

I. C. 6537

The East Texas Gravel Co.'s deposits are in old flood plains of the Trinity River. The material is typically fluvial, showing well-rounded, smooth-surfaced limestone gravel and angular siliceous sand. Prospecting in this region is largely confined to test pits and churn or bore holes. Many of these are negative or blanks. This illustrates the expense attendant on examining erratic deposits, or deposits which change in character within short distances. Flood plains follow the course of present or ancient streams. In the latter case later erosion has destroyed surface evidence of their presence except what remains in the topography resulting from terraces. Test pits are dug by hand 3 by 5 feet in sections where the walls will stand. When loose a cylindrical steel curbing 3 feet in diameter is used to protect the men. The small orange-peel bucket and the churn bailer are also used, the former above the permanent water line and the latter below the water line. The bailer itself is 8 inches in diameter fitted with a flap valve and a 3-pronged cutting edge. When operated by hand it should not be more than 2 feet long. This will require 8 men working 6 at a time. This type of bailer is also used on a cable-type well drill to take the place of the drill bit and rods. With the latter equipment 2 men can drill 70 to 80 feet in 10 hours.

The author cites the use of a rotary boring machine mounted on a small gasoline tractor which successfully bores a hole 16 inches in diameter in clay to a depth of 18 feet. This machine will bore at the rate of from 1 to 5 feet per minute in such material but did not prove successful in cutting gravel because of pebbles caving from the sides of the hole.

I. C. 6580

At Oxford, Michigan, the Ward Sand & Gravel Co. are operating in what is probably the remains of a ground moraine, the upper portion of which has been reworked by fluvioglacial waters which have removed most of the clay. Clay hills to the north are probably the remains of old terminal or recessional moraines from which glacial waters removed portions and deposited them on the ground moraine. This is indicated by the gradation from coarse gravel near these clay hills to fine sand on the southern side of the deposit. Below the permanent water line occasional beds of clay are found in the deposit showing little erosion by running water.

Prospecting and exploration was done by sinking test pits from 10 to 40 feet or to water level. Timber was used in some test holes in the form of 2-inch planks laid as cribbing. A churn drill was used to explore below the water level. The author expresses his opinion that "it is impossible to form an exact estimate of the value of the deposit until after it has been put in production."

Since this is probably a ground moraine topped by a shallow outwash plain, it is not surprising that analysis of test results might be puzzling. A knowledge of the geology would be a great help as a basis from which to interpret test-hole data, and with such knowledge a more comprehensive interpretation might be possible. Ground moraines, formed as they are beneath the ice, are heterogenous mixtures of material which present difficulties in analysis to the most experienced samplers.

I. C. 6581

The deposit at Clowdy, Tex., operated by the Dallas Washed & Screened Gravel Co. lies about 1½ miles from the present channel of the Trinity River.

Composed of material ranging from fine, angular siliceous sand to well-rounded limestone pebbles up to 3 inches in diameter, it is typical of fluvial material which has traveled a comparatively short distance. This particular deposit is probably the site of a previous channel of the river, as it cut its way across the coastal plain. The material deposited came from limestone ledges in the higher ground to the north and west. The Trinity River is now a silt-bearing stream and has not carried gravel for a considerable time. This accounts for the heavy clay mantle above the gravel at Clowdy. This is then a river-bed deposit formed in the coastal plain by a meandering river and later covered by silt or clay. When the river left this channel it frequently flooded and overran its banks, leaving the clay mantle as a flood-plain deposit.

In prospecting, this company used auger holes spaced at 300-foot intervals. These were sunk to water level. It will be noted that the overburden was clay and the gravel loose and without bowlders. This is most favorable to auger prospecting. These holes were drilled by hand at a cost of 15 cents per foot. They were used, however, only to outline the deposit and to determine the depth of overburden.

Next, test pits were sunk on 600-foot intervals to determine the character and thickness of the deposits. From the pits it was found that the permanent water level was from 4 to 8 feet above the bottom of the gravel, with the same variation below the top of the gravel.

An average of 7 feet of overburden was also disclosed. The shallow water and comparative thin gravel stratum (12 feet) indicated that drag lines would prove more economical equipment than dredges.

Some test pits were sunk by hand at a cost of $1.50 per foot and others by a 1-yard caterpillar crane at a cost of $1 per foot.

An attempt to use a 6-inch casing with a 5-inch bailer for testing below water level was unsuccessful where the material contained a considerable proportion of pebbles 1½ inch or larger in diameter, because of the slow progress incident to pebbles binding between bailer and casing. Possibly if the bailer had been smaller in diameter, better progress would have been made.

The report describes the methods employed in analyzing tests as follows:

> After the areas and thicknesses of the gravel deposits were determined, the volume of gravel was calculated and converted into tonnage by assuming a weight of 3,000 pounds per cubic yard. Screen analyses of the material from various test holes were then made and the average percentage of sand and gravel that should be recovered from the property was estimated.
>
> Two methods were used to obtain samples for screen analysis. When test holes were dug with the dragline, the overburden was cast on one side of the hole and the gravel on the opposite side; then an average sample was selected and shoveled into a sack. When test holes were dug by hand, samples were obtained above the water line by removing two shovelfuls of material from the side of the test hole. These samples were taken at regular depth intervals of 3 feet and all samples from a hole were mixed to obtain the average of the material above the water line. Samples from below the water line were removed from the center of the holes at the same depth intervals. The information obtained from these tests has been valuable in coordinating mining with market requirements due to the variance in both the quality and size of material in the deposits and the size and grading of material required by customers.
>
> Tests on all of the Trinity River gravel deposits have shown that the thickness of the strata, size and quality of material, and even the color of the gravel and sand change within short distances. The deposits are irregular in shape and change from gravel to sand, clay, or loam, in some instances in a distance of 10 feet or less.

Because of the erratic nature of this deposit, it would appear that at least occasional tests would be needed at shorter intervals than the 300 feet indicated.

I. C. 6582

The J. K. Davison & Bro. Co. operate ladder-type dredges in the Allegheny River near Pittsburgh, Pa. The deposits are river bars composed largely of large pebbles and bowlders brought down from Canada in a glacial stream of torrential proportions. The sands are siliceous and angular while the pebbles are well rounded. The absence of fine sand and silt except on top of the deposits and the low content of coarse sand indicate the fluvioglacial source. Recently finer sands and silts have been brought down from the erosion of sedimentary rock above and dropped on the original deposits, but these are thin and of minor importance. The reader is referred to U. S. Geological Survey Bulletin 430, pages 388 to 399, by C. W. Shaw, for a more detailed geologic description of these deposits.

The company has never spent much time in systematic prospecting. Prior to the erection of navigation dams in the river the bars were exposed to visual examination. The author admits, however, that mental notes of the location of these bars has not always proved accurate. Prospecting and exploration is confined to actual digging by the dredge.

I. C. 6592

The operations of the Seaboard Sand & Gravel Co. are confined to the moraine deposits on the north side of Long Island. These deposits are described by Nevin[20] as a terminal moraine left by an early ice sheet originating in Labrador. It is supposed that the southern half of the island is a second moraine left by a succeeding ice advance.

The author described the deposits as follows:

> The surficial deposits of Long Island are of glacial origin. They are composed almost entirely of sand and gravel, particularly along the north shore, but there are occasional beds of clay and bowlders. The clay varies in quantity and occurs at almost any depth in the deposits. The bowlders, however, are more systematic in their occurrence. In every case in the writer's experience, they are found at the surface, or within 8 or 10 feet of it. The overburden on Long Island consists of top soil, which never exceeds a depth of 2 feet and is not more than 4 or 5 inches thick on the property now being worked. Here the sand and gravel occur in well-defined strata, which vary in thickness from a few inches to several feet. These strata are more clearly defined near the surface where the gravel content is much heavier than deeper in the bed. With depth the layers disappear and the material seems to be largely sand with occasional pebbles in it. No clay has been encountered in the bank now being worked.

From the author's observation of occasional accumulations of clay and bowlders within the sand and gravel, one infers the moraine origin. He also states, however, that the sand and gravel occur in clearly defined strata near the surface. Since Nevin finds evidence of two moraine deposits, of which that on the north shore of the island is the older, it is probable that this deposit was subjected to considerable reworking by glacial waters and possibly wave action. This would account for the gradation near the surface with untouched clay accumulations below in the original moraine.

Prospecting this deposit was simple, as it occurs on a promontory exposed on three sides. Shallow test pits were dug, however, to confirm the visual inspection.

I. C. 6607

The deposit exploited by the Consolidated Rock Products Co., at Durbin, Calif., is a typical alluvial fan deposited by the San Gabriel River. There are no distinct layers of sand or gravel, but the material ranges coarser in size as the river canyon is approached. The permanent water level was determined at 110 feet below the surface. Drill-hole logs indicate a depth of deposit of over 300 feet.

Previous operators had thoroughly prospected the property but to confirm their results the company dug two 4 by 4 foot test pits to water level at a cost of $4 per foot. The material dug from these pits was tested and the information used to design a plant to handle the several gradations indicated.

The pits were timbered in 4-foot square-sets, and the material excavated between each set was piled separately on the surface. Five-hundred-pound samples were taken from each pile and first screened through a 3/8-inch screen to separate the sand from the gravel. The gravel was then put over a screen with 3-inch round perforations. The undersize from this screen represented gravel and the oversize that proportion that would require crushing. The

20 - Nevin, Chas. M., The Sand and Gravel Resources of New York: New York State Museum, Bull. 282, 1929, pp. 124-125.

gravel sample was then separated in the various commercial sizes to determine their ratio of occurrence.

SUMMARY

In this report the author has endeavored to point out the necessity for proper prospecting and exploration of a sand and gravel deposit before exploitation begins, the reasons for such necessity, the information to be gained, and the application of such information in valuing the material.

Typical examples of methods employed by operating companies have been cited and costs given when available.

It is hoped that the information contained herein will be of assistance and guidance to operators contemplating the opening of new deposits and that it will induce them to ascertain thoroughly all available facts before investing capital, in order that such investment be founded on facts rather than hopes. In this way waste may be prevented and unwise exploitation avoided.

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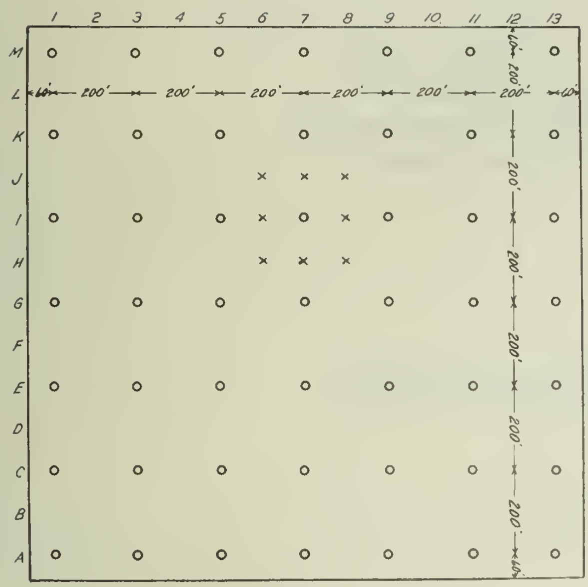

Figure10.- Drill-hole plan

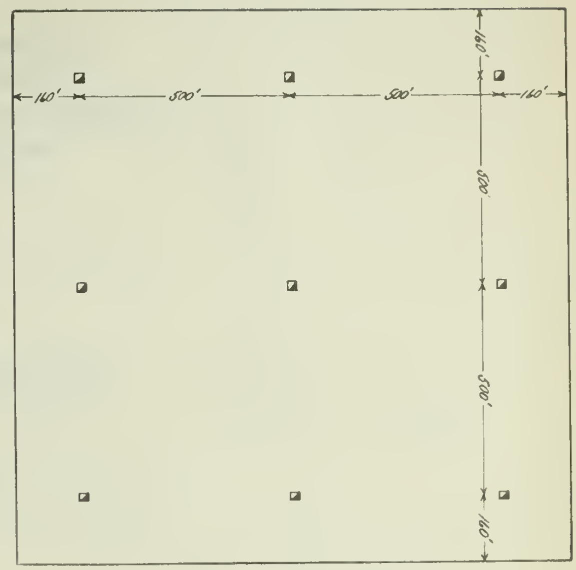

Figure11.- Test-pit plan

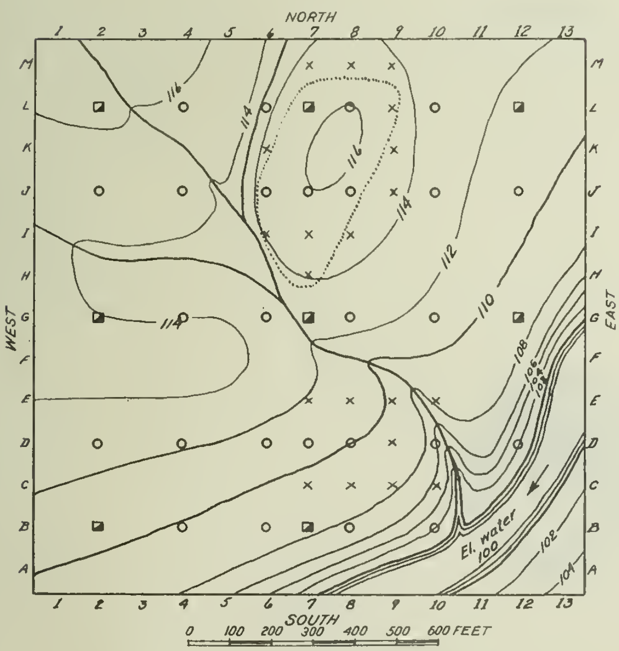

Figure12.- Contour map

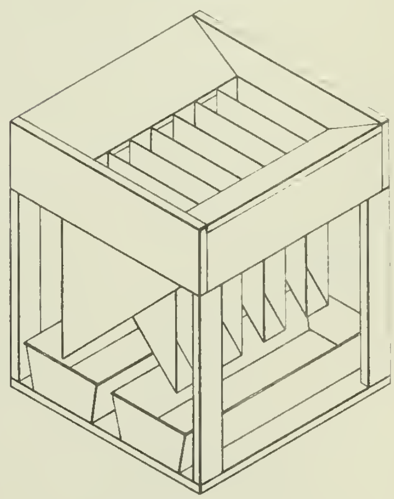

Figure 13.- Sample splitter

I. C. 6669 September, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

GEOPHYSICAL ABSTRACTS

NO. XLI

BY

Frederick W. Lee

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

GEOPHYSICAL ABSTRACTS[1]
No. 41

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2 Senior physicist, U. S. Bureau of Mines.

1. GRAVITATIONAL METHODS

(967) TERRAIN AND TOPOGRAPHICAL EFFECTS IN GRAVIMETRIC SURVEYS

By W. G. G. Cooper

The Mining Magazine, London, vol. 46, No. 5, 1932, pp. 282-285.

In this article the author suggests a method of computing terrain and topographical effects in survey work with the Eötvös torsion balance. The suggestion embodies the use of slide rule, graphical work, and planimeter, and it is claimed that the accuracy of these instruments is equal to or greater than that of the assumptions made regarding density of material, etc. Levels are taken along radial lines (8 in normal country) at any convenient distances, giving a fair representation of the ground.

The theory of the method is discussed.--W. Ayvazoglou.

(968) SURVEYS WITH THE TORSION BALANCE AND THE MAGNETOMETER IN EASTERN CANADA

By A. H. Miller

The Journal of the Royal Astronomical Society of Canada, Toronto, vol. 26, No. 1, 1932, pp. 1-16.

The author discusses the results of the torsion balance and magnetometer surveys carried out in Ontario over the Hull-Gloucester fault at Leitrim, the Hazeldean fault, the Caldwell pyrite deposit, and the dunite intrusive at Thetford, Quebec.

The results are represented in a series of plans and diagrams.--W. Ayvazoglou.

(969) TEKTONISCHE UND GEOPHYSIKALISCHE PHÄNOMÄNE IN DER FERGHANREGION, ZENTRALASIEN

(TECTONIC AND GEOPHYSICAL PHENOMENA IN THE REGION OF FERGHANA, CENTRAL ASIA)

By Franz Kossmat

Zeitschrift der Deutschen Geologischen Gesellschaft, Berlin, vol. 84, No. 2, 1932, pp. 84-94.

Geophysical phenomena in the region of Ferghana are discussed in Chapter IV of this article. The region is characterized by great negative gravity anomalies. A few especially typical values are given in a table. --W. Ayvazoglou.

(970) LA PROSPECTION GRAVIMETRIQUE DANS LES REGIONS PETROLIFÈRES

(GRAVIMETRICAL PROSPECTING IN THE OIL-BEARING REGIONS)

Editorial note

Le Génie Civil, Paris, vol. 100, No. 14, 1932, p. 355.

The note relates to the papers presented by Gornich and Reich during the general meeting of the German Geological Society, held from September 14 to 17, 1931 (see Geophys. Abs. 34, p. 372).--W. Ayvazoglou.

2. <u>MAGNETIC METHODS</u>

(971) AN IMPROVED METHOD FOR THE COMPARISON OF SMALL MAGNETIC SUSCEPTIBILITIES

By R. A. Fereday

Proceeding of the Physical Society, London, vol. 43, part 4, No. 239, 1931, pp. 383-393.

In an earlier communication (Proc. Phys. Soc., vol. 42, 1930, p. 251) an account was given of a new method, of the nonuniform field type, for the comparison of small magnetic susceptibilities.

In this article the author continues his investigations and shows that a magnetic field, especially suitable for determination of relative susceptibility by an improved nonuniform field method, can be produced by an electromagnet whose pole pieces are respectively plane and spherically concave.- W. Ayvazoglou.

(972) THE MAGNETOMETER IN ILLINOIS

By Perry S. McClure

Transactions of the Illinois State Academy of Science, Springfield, vol. 24, No. 2, 1931, pp. 341-349.

A description is given of magnetometer surveys carried out in Eastern Illinois where the geologic structure is well known. The entire survey covers an area of 3,240 square miles. Some 1,620 magnetometer stations were established in these areas to measure the relative vertical magnetic intensity.

This report points out some of the principal anomalies of the earth's magnetic field in Illinois as revealed by measurement of the vertical component at numerous points and describes the effect of faults on the vertical magnetic intensity.

A discussion under the following headings is given: (1) Earth's magnetic field, (2) history of magnetic exploration, (3) magnetic field balance, (4) interpretation of magnetic anomalies, (5) magnetic anomalies in Illinois, and (6) tracing faults with the magnetometer.

The author concludes that the magnetometer has two principal uses in Illinois. First, in regional reconnaissance surveys it is quite possible to pick up structural trends, and second, in doing detailed work it will indicate the position of faults.

(973) CATALOGUE OF MAGNETIC DETERMINATIONS IN THE U.S.S.R. AND IN ADJACENT COUNTRIES FROM 1556 TO 1926

By B. P. Weinberg

Issued by the Central Geophysical Observatory in Leningrad, 1932, pp. 216-297.

This is the second part of the catalogue issued by the Central Geophysical Observatory in Leningrad (see Geophys. Abs. 11).

The catalogue is supplemented by three charts reduced to the epoch 1925:

1. Chart of isogamic lines.
2. Chart of isoclinal lines.
3. Chart of isodynamic lines.

Explanations to the charts are given in the Russian and English languages.--W. Ayvazoglou.

(974) PRELIMINARY SUMMARY OF DATA ON THE PRESENT DISTRIBUTION OF MAGNETIC DECLINATION WITHIN THE ARCTIC ZONE

By Boris P. Weinberg

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, vol. 36, No. 4, 1931, pp. 273-278.

Owing to the fact that the determinations of the magnetic elements in the arctic regions are irregularly distributed over the territory, as well as in time, so that it is difficult to find their distribution for a definite epoch in a sufficiently trustworthy manner, the author tries to solve this problem by using the method of successive approximations relating to an element $\mathcal{E}$ and to its secular variation (annual change) $\mathcal{E}'$.

The method summarized in this article has been developed as the results of Weinberg's experience in reducing to 1925 all the magnetic determinations in the U.S.S.R. and adjacent countries. Data on the distribution of magnetic declination within the arctic zone is summarized in figures and a table.--W. Ayvazoglou.

(975) TERRESTRIAL-MAGNETIC ACTIVITY AND ITS RELATION TO SOLAR PHENOMENA

By J. Bartels

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, vol. 37, No. 1, March, 1932, pp. 1-52.

A homogeneous series of monthly means of terrestrial-magnetic activity for the years 1872 to 1930 is derived and extended backward, in annual means, to 1835. The annual variation of magnetic activity and of the relative sun-spot numbers is discussed by means of new tests for periods. Only the semi-annual wave in magnetic activity is recognized as physically significant. Its maxima prefer the times when the sun is in the celestial equator, and not, as has been suggested, the times when the sun's axis is most inclined toward the ecliptic. This view is supported by tests using the harmonic dial and the probable-error circle, and by several independent considerations.

The close relations between sun-spot numbers and terrestrial-magnetic activity in the annual and monthly means are discussed. Some general statistical aspects are given for the treatment of the correlation between such series with after effects, for which both solar activity and terrestrial-magnetic activity are typical. The homogeneity of the whole available series for relative sun-spot numbers and for areas of sun-spots and faculae is tested; some inhomogeneities are found, apart from a general lag of terrestrial-magnetic activity that has occurred in some sun-spot-cycles. A break in the homogeneity of the international magnetic character figures in recent years is discovered.

The individual 27-day recurrences in terrestrial-magnetic activity during 1906-1931, and their relations to solar activity are discussed with the help of a graphical day-by-day record. They indicate the existence of persistent activity areas on the sun's surface, called M regions, which, in many cases, can not be coordinated to such solar phenomena as are observable by direct astrophysical methods. This holds in particular for the new solar indices which are available for the years 1928-1930, and which are found so closely correlated to sun-spot numbers, that they fail to improve the correlation between solar activity and terrestrial-magnetic activity. Observations of terrestrial-magnetic activity yield therefore not only information about geophysical influences of such solar phenomena that may be traced in astrophysical observations, but supplement these direct observations themselves--Author's abstract.

(976) ZAMERENI A VYPOCET MAGNETICKE DEKLINACE (IN BOHEMIAN)

(OBSERVATION AND CALCULATION OF MAGNETIC DECLINATION)

By Fr. Cechura

Hornicky Vestnik (Mining Journal), Prague, 1931, pp. 1-18.

After a brief introduction in which the author indicates the practical importance of knowing the correct magnetic declination, a description of the Schmidt and Neumayer mirror-declinometer and of Hildebrand's magnetic theodolite is given.

The process of measurements is discussed. Checking of the results in order to avoid errors in calculations is explained.--W. Ayvazoglou.

(977) A TYPICAL CASE OF VARIABILITY OF QUIET-DAY DIURNAL VARIATION IN TERRESTRIAL MAGNETISM AND EARTH CURRENTS AT WATHEROO

By J. Bartels and W. J. Rooney

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, vol. 37, No. 1, March, 1932, pp. 53-55.

The small magnetic diurnal variation recorded at the Watherloo Observatory, Watheroo, Western Australia, on two successive magnetically very quiet days, February 3 and 4, 1929, was noteworthy. This case is discussed as typical for the variability of the quiet-day diurnal variation. A significant parallelism is found with the diurnal variation of earth currents.--Authors' abstract.

(978) TEST-DEFLECTIONS FOR VARIOMETERS AND MAGNETOGRAPHS

By George Hartnell

Terrestrial Magnetism and Atmospheric Electricity, Baltimore, vol. 37, No. 1, March, 1932, pp. 63-78.

This is the continuation of the article published under the same title in "Terrestrial Magnetism and Atmospheric Electricity, vol. 36, No. 4, 1931, pp. 279-296. Author's abstract of the article is given in Geophysical Abstracts 37, May, 1932, p. 442.--W. Ayvazoglou.

(979) THE AGINCOURT SCHUSTER-SMITH COIL-MAGNETOMETER

By W. E. W. Jackson

Terrestrial Magnetism and Atmospheric Electricity, Baltimore,
vol. 37, No. 1, March, 1932, pp. 79-82.

In this article the author describes the function of a Schuster-Smith coil-magnetometer installed in August, 1931, in the magnetic observatory at Agincourt.

The principle of the instrument is given briefly.

A table shows the results of observations made on August 12, 1931, and the results of comparison with the magnetograph, the values of which are deduced from observations with Elliott magnetometer No. 98 reduced to the International Magnetic Standard of the Department of Terrestrial Magnetism of the Carnegie Institution.--W. Ayvazoglou.

(980) MAGNETIC CONVERGENCE MAPPING

By W. I. Ingham

The Mines Magazine, Golden, Colorado, vol. 22, No. 7, 1932, pp. 7-8.

This paper deals with a method of mapping and interpretation used by the writer in some areas in Texas, while making magnetic surveys with the Schmidt vertical-intensity magnetometer.

Preparation of the magnetic convergence maps is explained.--W. Ayvazoglou.

(981) (THEORY AND EXPERIMENTS CONCERNING A NEW COMPENSATED MAGNETOMETER SYSTEM

By C. A. Heiland and W. E. Pugh

The American Institute of Mining and Metallurgical Engineers,
Technical Publication 483,
1932, 42 pp.

Heiland's summary of the paper reads as follows:

The angle of deflection in magnetic-intensity variometers indicates a balance between the magnetic force component to be measured and another known force, which is either (1) of a magnetic or (2) of a mechanical nature (torsion of wires, gravity moment). Only in the first case can the influence of temperature be avoided, as the temperature coefficient of the needle cancels; in the second case, a compensation for temperature is possible, as its magnetic effect can be partly neutralized by an opposing "mechanical" effect. Although the magnetic effect is usually reduced as much as possible by using

systems of great moment, as a rule it remains greater than the mechanical effect. Then additional compensation is required, which may be of a magnetic (auxiliary magnet) or of a mechanical nature (temperature affecting torsion of suspensions or the position of the center of gravity, etc.) In vertical magnetometers, the last type of compensation is used mostly. In the old type of Schmidt balance it was accomplished by shifting the blades; in the new system, by suitably arranged temperature and latitude spindles.

The theory of the temperature effect on a magnetic system is given in two parts. First, the magnetic temperature coefficient only is considered, in its effect on scale value and reading. The magnetic T.C. of the scale value is negligible ($=\mu \varepsilon$), μ being the temperature coefficient of the magnetic moment. The magnetic T.C. of the reading is very large (T.C. = $-\mu Z$), proving that the T.C. depends on the latitude.

Second, the combined "mechanical" and "magnetic" effects are discussed. For the T.C. of the scale value: T.C. = $(\delta + \mu)\ \varepsilon$, δ being the expansion coefficient of steel, the T.C. of the scale value is again negligible. For the temperature coefficient of the reading, T.C. = $- Z_0\ (p + \mu)$, Z_0 being the vertical intensity for which the system is adjusted, p representing the "mechanical" temperature coefficient; it may be made negative and equal to μ by a suitable arrangement of the masses in the system.

There is an influence of latitude upon a compensated system, as Zp remains practically constant, but $- Z\mu$ changes. The temperature coefficient, therefore, increases with an approach of the equator. The change is not very great; a system compensated for Golden may be used without readjustment practically throughout the United States.

If the T.C. is expressed in gammas, changes in scale value should not affect it; however, there may be such an influence, if a change in the ε-screw changes the lateral position of the center of gravity. In the magnetic system investigated, the T.C. increases with scale value; the increase is small for high, great for low magnetic latitudes; yet, in both cases close to the observational error. The theory shows also that, if the T.C. is not determined near the reading 20, the deflection of the system enters, as it changes Z; however, the influence is small.

Computations of T.C. based upon the theory as outlined have been checked by the experiments with an accuracy of $\pm$ 0.3 γ and less.

The proposed experimental work required a number of preliminary investigations, such as the determination of the magnetic moment of the system and of its temperature coefficient, investigations into the most suitable procedures for changing temperatures, scale values, and latitudes. It was found that the simultaneous recording of the

magnetic variations was required, which in turn necessitated a fairly constant temperature in the recording room. For the accurate determinations of temperature coefficients, arrangements had to be made to use always the same temperature gradient in order to obtain comparable values for different scale values and latitudes. The amount of lag of the reading behind the recorded temperature was accurately determined for the gradients employed.

Numerous temperature curves were recorded for normal conditions at Golden, for different latitudes varying between vertical intensities near the magnetic equator and intensities exceeding Z at the pole, and, finally, for different scale values.

The results are shown graphically, and are in close accord with the theoretical deductions. They show a decrease of the temperature coefficient with increase in latitude.

It is concluded that the new magnetic system gives reliable results for a great variety of latitude, scale value, and temperature conditions. The derivation of the complete theory makes it possible to determine accurately the influence of any changes in these factors upon the T.C., and to calculate the change in T.C. for any changes in the distribution of the masses of a magnetic system.--C. A. Heiland.

3. SEISMIC METHODS

(982) ON THE VELOCITY OF THE P-WAVE, WITH SPECIAL REFERENCE TO THE DISCONTINUITY RECENTLY SUGGESTED BY DR. H. JEFFREYS

By Hiroshi Kawasumi

Japanese Journal of Astronomy and Geophysics, Tokyo, vol. 9, No. 1, 1931, pp. 15-22.

The velocity of the P wave in the substratum up to the Wiechert discontinuity was re-estimated by assuming the Rudski-Wiechert formula $v = a - br^2$, using the mean travel times of 85 earthquakes recently obtained by H. Jeffreys. The depth of the Wiechert discontinuity was also determined. The results coincided within almost 1 per cent with the previous determination of the author from a near earthquake, and the general accuracy has increased. This good coincidence and the smallness of the residuals show that the velocity formula above determined is sufficient for the present so far as it concerns the accuracy of determining the travel time. The depth of the Wiechert discontinuity came out very near that of previous determination. Some light has been thrown on the smallness of the focal depth of large earthquakes of normal origins.--Author's abstract.

(983) CHARACTERISTICS OF THE OSCILLOGRAPH GALVONOMETER: SOME PRACTICAL CONSIDERATIONS IN THE DESIGN AND APPLICATION OF THE OSCILLOGRAPH-GALVANOMETER VIBRATOR

By V. S. Thomander

Journal of the Franklin Institute, Philadelphia, vol. 213, No. 1, 1932, pp. 41-55.

The paper describes the mechanical characteristics of the vibrator for any periodic phenomena, so that the mechanical error due to the vibrator may be eliminated from the oscillogram and the true phenomena determined. The paper establishes the best damping to use, when considered from the standpoint of vibrator response, lag, and transient characteristics. There is a method given to determine the vibrator resonance and damping, that may easily be applied in the field. Curves, from which all the above-mentioned may be obtained, are supplied.--Author's abstract.

(984) A SIMPLE SUSPENDED MIRROR SEISMOGRAPH

By Benjamin Allen Wooten

Science, New York, vol. 75, No. 1932, 1932, pp. 82-83.

The apparatus consists essentially of a light mirror about 3 mm in diameter hung by means of two silk fibers in an F frame after the manner of Darwin. The motion of the mirror is traced on a photographic film, which is attached to a rotating drum in the usual manner.

The apparatus has been used thus far in the study of earth tremors and disturbances of a minor nature. It detects with ease the footsteps of a person 100 yards from the building. It records the passing of a street car or an automobile half a mile away. According to the author this apparatus is particularly well adapted to the study of tremors which do not penetrate the earth very deeply. It is light, simple, inexpensive, and easily portable; and it can be set up with a minimum of adjustment.--W. Ayvazoglou.

(985) A NEW VERTICAL SEISMOGRAPH

By Hugo Benioff

Bulletin of the Seismological Society of America, Stanford University, Calif., vol. 22, No. 2, 1932, pp. 155-169.

The vertical seismograph described in this paper makes use of a mechanical system of short period in combination with an electromechanical "transducer" of high sensitivity coupled to a recording galvanometer. (According to the footnote an electromechanical transducer is a device actuated by power from a mechanical system and supplying power to an electrical system, or vice versa.) In the Galitzin seismograph the transducer consists of a

coil which moves in a magnetic field in such a manner as to generate electromotive force proportional to the velocity of the inertia reactor. The present transducer embodies a modification of the common telephone-receiver principle and is also of the velocity type.

The mechanical system of the seismograph is described and the theories of the electromechanical transducer and of the seismograph are given.--W. Ayvazoglou.

(986) REFLECTION METHOD IN SEISMIC PROSPECTING

By H. M. Rutherford

The American Institute of Mining and Metallurgical Engineers, New York, Technical Publication 486, 1932, 22 pp.

The purpose of the present paper is to present the method of reflections in the mapping of geologic structure and also to give some indications of its limitations.

The problems connected with the reflection method are summarized by the author in part as:

1. General design of instruments.
2. Identification of the reflection on the seismogram.
3. The correlation of the reflection with the reflecting bed.
4. Methods of computation.

The relationship between the time-distance curve for refracted waves and that for reflected waves is developed. General formula for reflections is derived. A discussion is given on the (1) path of reflected ray in case of several beds, (2) the average velocity, (3) the surface (or weathering) correction, (4) reduction to mean datum, and (5) curved paths.

The last part of the article deals with the examples of reflections as shown in graphs and tables.

The author concludes:

> It is pertinent to point out that seismic methods of prospecting have not reached, and may never reach, a stage where the calculations are merely a matter of routine. Geological conditions are not the same all over the country. They vary considerably even in any one region. Methods that might give close approximations to true conditions in one territory may be found to give very great errors if used in another.
>
> In general, the results presented in this paper are the outgrowth of several years of experience in seismic prospecting. Much work remains to be done, both experimentally and in methods of interpretation of data.--W. Ayvazoglou.

(987) POSSIBILITY OF FREE OSCILLATIONS OF STRATA EXCITED BY SEISMIC WAVES

By K. Sezawa and K. Kanai

Bulletin of the Earthquake Research Institute, Tokyo,
vol. 10, No. 2, 1932, pp. 273-298.

The present paper continues as a report of the authors' research work to determine the possible range of free oscillations of strata due to a seismic disturbance (see Geophys. Abs. 38, p. 477). This paper deals with a case where primary seismic waves are incident upwards normally to twofold stratified layers resting on the surface of a semi-infinite body.

Thicknesses, densities, and elastic constants of the two layers and of the bottom medium, obtained from the results of investigations made by Matuzawa and Imamura, and used by the authors for their mathematical discussion, are as follows:

Layer	Thickness	Density	V for P waves	V for S waves
Upper layer (granite)	H'-H = 20 km	$\rho'' = 2.7$	5.0 km/sec.	3.15 km/sec.
Second layer (basaltic)	H = 30 km	$\rho' = 3.0$	6.1 km/sec.	3.70 km/sec.
Bottom medium (ultrabasis)	∞	$\rho = 3.5$	7.5 km/sec.	4.45 km/sec.

A few numerical examples are shown in figures.--W. Ayvazoglou.

(988) AMPLITUDES OF P AND S WAVES AT DIFFERENT FOCAL DISTANCES

By K. Sezawa and K. Kanai

Bulletin of the Earthquake Research Institute, Tokyo,
vol. 10, No. 2, 1932, pp. 299-335.

This paper is concerned with P and S waves of equal periods generated from a point in a visco-elastic body and transmitted in all directions toward infinity. To determine the amplitudes of the waves in all probable cases, the authors considered the generation of the waves in a certain distribution in different azimuths as well as different latitudes.

From the results of their theoretical investigations, the authors conclude that, at any rate, the abnormal smallness of the amplitudes of P waves can not be ascertained directly from the nature of the wave propagation.--W. Ayvazoglou.

(989) ON THE EXPRESSIONS OF THE DEFORMATION OF A SEMI-INFINITE ELASTIC BODY DUE TO THE TEMPERATURE VARIATION

By G. Nishimura

Bulletin of the Earthquake Research Institute, Tokyo,
vol. 10, No. 2, 1932, pp. 335-351.

Nishimura's investigations presented in this article are closely connected with those published by him in the Bulletin of the Earthquake Research Institute, vol. 8, No. 2, 1930, pp. 91-143, under the title "The effect of temperature distribution on the deformation of a semi-infinite elastic body." (See Geophys. Abs. 17, p. 20.)

Nishimura studied the deformation of the solid due to temperature distribution and variation satisfying all the conditions of the theory of heat conduction.

The two following cases were examined: The case where the inertia of solid was neglected, and that where the inertia effect was taken into account. In both cases the resulting equations contained the term of density, in spite of the fact that the inertia term was neglected in one case and not in the other.--W. Ayvazoglou.

(990) INVESTIGATION ON THE DEFORMATION OF THE EARTH'S CRUST IN THE TANGO DISTRICT INCIDENT TO THE TANGO EARTHQUAKE OF 1927

By Chūji Tsuboi

Bulletin of the Earthquake Research Institute, Tokyo,
vol. 10, No. 2, 1932, pp. 411-435.

Previous investigations of the deformation of the earth's crust in the Tango district brought about by the Tango earthquake of 1927 have been made by the author (see Geophys. Abs. 18 and 34). In this article Tsuboi continues his investigation and discusses the results of the last precise levelings carried out by the Land Survey Department of the Imperial Army.--W. Ayvazoglou.

(991) INVESTIGATION OF THE DEFORMATION OF THE EARTH'S CRUST IN IDU PENINSULA CONNECTED WITH THE IDU EARTHQUAKE OF NOVEMBER 26, 1930

By Chūji Tsuboi

Bulletin of the Earthquake Research Institute, Tokyo,
vol. 10, No. 2, 1932, pp. 335-352.

In his previous paper (Bulletin, Earthquake Research Institute, Tokyo, vol. 9, 1931, p. 271) Tsuboi compares the old and new precise levelings made

around the peninsula and discussed the results. In the present paper he discusses the differences between the old and new triangulation over the area. The movement of each triangulation point is established. The deformation of the earth's crust is investigated on the basis of the results.--W. Ayvazoglou.

(992) SEISMOMETRICAL REPORT

By N. Nasu and Ch. Yasuda

Bulletin of the Earthquake Research Institute, Tokyo, vol. 10, No. 2, 1932, pp. 492-498.

The report gives:

1. Sensible earthquakes in Tokyo for the period October 1 to December 31, 1931.

2. Important distant earthquakes as observed in Tokyo.

3. A map of distribution of the earthquakes that originated within a distance of 160 kilometers from Tokyo.

For previous report see Geophysical Abstracts 34.--W. Ayvazoglou.

(993) TWO-COMPONENT VIBROGRAPH

American Askania Corporation

Instruments, Pittsburgh, Pa., vol. 5, No. 1, 1932, p. 21.

A special feature of this instrument, designed for the study of mechanical vibrations, is that the vibrations are converted into visible wave forms by a mechanical optical arrangement, rather than by use of a cathode-ray oscillograph.

To determine the extent and direction of the vibrations, the movement of the ground is broken up into its three spatial components. The three pendulum weights in the instrument are therefore so arranged that they are affected only by forces acting in one of three directions at right angles to each other.--W. Ayvazoglou.

(994) TRI-DIMENSIONAL VIBROGRAPH

By the Vibration Specialty Co.

Instruments, Pittsburgh, Pa., vol. 5, No. 5, 1932, p. 136.

The vibrograph is a form of seismograph which indicates and records components of vibration simultaneously in a vertical and two horizontal

directions. The instrument comprises a heavy cubic mass suspended in a frame by a system of springs, and means for indicating and recording relative motion between the mass and the frame. When placed upon any vibrating body the frame vibrates with the body while the mass due to its inertia tends to remain at rest.--W. Ayvazoglou.

(995) THE JAPANESE EARTHQUAKE OF MARCH 29, 1928, AND THE PROBLEM OF DEPTH OF FOCUS

By V. C. Stechschulte

Bulletin of the Seismological Society of America, Stanford University, California, vol. 22, No. 2, 1932, pp. 81-137.

A detailed investigation of the Japanese earthquake of March 29, 1928, to which Wadati, on the basis of the S-P intervals at the Japanese stations, had assigned great focal depth, has been made in order to see if the records at all distances show the expected effects of that depth.

1. The S-P, P, and S travel-time curves show systematic, graduated departures from the "average" curves.

2. These same curves have an inflection point in the vicinity of $\triangle = 10^o.5$.

3. The two pairs of singly reflected waves, pP and PR, sS and SR, were recognized, and tables of observed travel times for the pP and sS waves are here presented for the first time.

4. Tables of travel times for other phases, P^1, S_cS, S_cP_cS, $S_cP_cP_cS$, were drawn up and compared with the times given in the Macelwane tables.

The evidence drawn from the foregoing is all consistent in pointing to extraordinary depth of focus. To obtain a numerical evaluation of that depth, a new method was devised utilizing the travel-time curves for pP and P. The depth was thus calculated as 410 ± 30 kilometers. This same method also supplied means for deriving the time of occurrence and for improving the approximate location of the epicenter.

The coordinates of the epicenter are: 31° 45' N., 138° 12' E.

The time of occurrence was determined as $5^h\ 06^m\ 03^s$ G.M.T., March 29, 1928.

Further check on the focal depth of 410 milometers is afforded by the "reduced" travel times which have been calculated from the data of the present earthquake, and which are found to agree very satisfactorily with the times for shocks at or near the surface as given by various workers.

An alphabetical list of the Japanese stations, together with the arrival times of P and the S-P intervals as given by Wadati (On shallow and deep earthquakes: Geophys. Mag., Tokyo, vols. 1 and 2, 1929), is presented. This is followed by a list of the other stations from which data were available. Bibliography is added.--Author's abstract.

(996) THE EARTHQUAKE IN SANTA MONICA BAY, CALIFORNIA, ON AUGUST 30, 1930

By B. Gutenberg, C. F. Richter, and H. O. Wood

Bulletin of the Seismological Society of American, Stanford University, California, vol. 22, No. 2, 1932, pp. 138-154.

The occurrence of a moderately strong local earthquake in Santa Monica Bay, Calif., on August 30, 1930, is discussed and its epicenter and origin-time determined as follows: $\phi = 33^\circ 57'$ N., $\lambda = 118^\circ 38'$ W., $0 = 4 : 40 : 36.0$ p.m., P.S.T. ($0^h 40^m 36^s.0$, G.C.T., August 31, 1930).

The maximum intensity observed was VIII of the Modified Mercalli Intensity Scale of 1931, and the shock was perceptible at distances up to about 160 kilometers (100 miles) from the epicenter. The distribution of apparent intensity exhibits numerous irregularities which are discussed in the text.

The depth of origin can not be determined accurately, but the data are consistent with a depth of 10 to 15 kilometers (6 to 9 miles). A significantly greater depth is not consistent with the data.

The surface and subsurface geology of the region are discussed briefly.--Authors' abstract.

(997) TRENDS IN GEOPHYSICAL PROSPECTING WITH EXPLOSIVES

By Roland F. Beers

The Explosive Engineer, Wilmington, vol. 9, No. 8, 1931, pp. 288-289.

In selecting explosives for use in seismic prospecting, the author examines important characteristics such as strength, density, rate of detonation, water resistance, consistency, resistance to freezing, safety, and cost per unit of explosive energy.

The efficiency of personnel used in seismic prospecting is emphasized.--W. Ayvazoglou.

4. ELECTRICAL METHODS

(998) GEOPHYSICAL PROSPECTING. NEW SCHLUMBERGER METHOD

By A. Montgomery

Industrial Australian and Mining Standard, Melbourne, vol. 87, No. 2231, 1932, p. 144.

In this article Montgomery discusses the new electrical method of examining ground passed through by boreholes, as described by C. and M. Schlumberger and E. Leonardon in Technical Publication 462 of the American Institute of Mining and Metallurgical Engineers (see Geophys. Abs. 35, p. 392).--W. Ayvazoglou.

(999) ELECTRICAL PROSPECTING IN CANADA

By J. J. Brousse

Engineering and Mining Journal, New York, vol. 133, No. 6, 1932, pp. 337-338.

The purpose of this paper is to illustrate, by two examples of practical surveys, the use of the self-potential method in ore exploration.

The author discusses briefly the conditions under which the phenomenon of spontaneous polarization occurs, and the reasons why it is one of the most reliable and rapid techniques in the explorations for ore.

The electrical surveys of the F orebody of the Amulet Mines, in the Rouyn district of Quebec, and of the Brunschwig orebody near Hope, B. C., were examined.

In both cases the electrical conclusions agreed satisfactorily with the geological information.--W. Ayvazoglou.

(1000) EARTH RESISTIVITY SURVEY IN ILLINOIS

By M. King Hubbert

Engineering and Mining Journal, New York, vol. 133, No. 3, 1932, pp. 142-143.

Three types of problems were investigated by the earth-resistivity methods specified: (1) Faults in Paleozoic sediments, (2) gravel deposits and ground-water supply in glacial drift, and (3) anticlines buried beneath drift.

Apparatus used was the megger ground tester.

It was found that faults could be discovered, especially if the outcrops on the opposite side were different or if a considerable shear zone were associated with the fault plane. Major gravel deposits having fairly sharp boundaries could be detected readily because of their relatively higher specific resistivity. Moreover, the one buried anticline tested gave a pronounced anomaly, which coincided with the anticlinal axis as indepently determined from well-log data. Attempts at precise depth determinations have not as yet been successful.--W. Ayvazoglou.

(1001) OBSERVATION AND DEDUCTION APPLICABLE TO MEASUREMENT OF ELECTRICAL RESISTIVITY OF LARGE VOLUMES OF EARTH IN PLACE

By Carl Henry Knaebel

Bulletin of the Michigan College of Mining and Technology, Houghton, vol. 5, No. 2, January, 1932, 31 pp.

In this paper Knaebel discusses and developes mathematically the solutions for some of the simpler problems involved in that type of electrical prospecting known as the "surface potential method." The number of such problems is of course great, and the author expresses the hope that the results for those treated in this thesis will be of interest to the engineer and that the methods of attack explained here will aid the student of geophysics in solving many of the more difficult problems.

The solution of problems of current distribution and current flow in infinite conductors can, according to the author, seldom be found in treatises on current electricity, which deal chiefly with electrical machines and circuits. Thus in the present work Knaebel found it necessary to resort to the study of similar problems in the theory of electrostatics and then to transfer the solutions bodily to the current-flow problems.

The headings of the article are as follows:

Distribution of current in an infinite homogeneous medium.
Distribution of potential in an infinite homogeneous medium.
Potential distribution in the case of two layers.
Interpretation of field data.--W. Ayvazoglou.

(1002) GEOPHYSICAL PROSPECTING METHODS

Editorial note

The Miner, Vancouver, vol. 5, No. 7, 1932, pp. 213-214

This paper summarizes E. G. Leonardon's address on Schlumberger's methods of prospection, according to the following outline:

1. Exploration for conductive orebodies.
2. Stratigraphical and tectonical studies.
3. Dam site studies; mapping of buried valleys.
4. Measurements of resistivities in drill holes; electrical correlations.
5. Preliminary discussion of problems, examination of field data, test surveys.

The address was concluded by discussing the electrical measurement of resistivity and porosity inside of open drill holes by an insulated cable lowered in the hole to obtain the electrical characteristics of the rocks in situ.--W. Ayvazoglou.

5. RADIOACTIVE METHODS

(1003) ÜBER DEN LUFTDRUCKKOEFFICIENTEN DER HARTEN ULTRASTRAHLUNG

(ON THE AIR-PRESSURE COEFFICIENT OF HARD COSMIC RADIATION)

By W. Messerschmidt and W. S. Pforte

Zeitschrift für Physik, Berlin, vol. 73, No. 9/10, 1932, pp. 677-680.

The mean value of the air-pressure coefficient of hard cosmic radiation (protected from all sides by 10 cm thick lead plates) was found to be very constant during a period of measurements of half a year; the same was found for the intensity of radiation reduced to the height of the barometer equal to 750 mm Hg. During low barometric pressures of short duration, deviations from the mean air-pressure coefficient were observed.--Authors' abstract translated by W. Ayvazoglou.

(1004) COSMIC-RAY IONIZATION AND ELECTROSCOPE-CONSTANTS AS A FUNCTION OF PRESSURE

By Robert A. Millikan

The Physical Review, Minneapolis, vol. 39, No. 3, 1932, pp. 397-403.

1. The residual ionization in an electroscope at infinite depth in water, that is, its zero reading, is found to be an inverse function of the pressure. Thus, in a particular electroscope the zero at 1 atmosphere was 5.13 ions cm^3/sec., while at 30.1 it had fallen to 1.2 ions cm^3/sec.

2. Also, when in this electroscope the pressure was changed from 1 atmosphere to 30.1 atmospheres the observed ionization current rose but 13.80-fold, which multiplying factor was found to be the same for gamma rays and for cosmic rays.

3. Both of these pressure effects are shown to be due to lack of saturation in high-pressure electroscopes, as first explained in Nature of October 3, 1931, by Bowen and the author.

4. From the multiplying factor found in (2) in the measured ionization at Pasadena in this 30-atmosphere, high-pressure electroscope, the number of cosmic-ray ions at 1 atmosphere (24° C., 74 cm pressure) in this electroscope at Pasadena is found to be fairly accurately 2.63 ions cm^3/sec.

5. The sea-level value of the ionization in this electroscope is 2.48 ions cm^3/sec.--Author's abstract.

6. GEOTHERMAL METHODS.

(1005) RECENT GEOTHERMAL MEASUREMENTS IN THE MICHIGAN COPPER DISTRICT

By James Fischer, L. R. Ingersoll, and Harry Vivian

The American Institute of Mining and Metallurgical Engineers, New York, Technical Publication No. 481, 1932, 11 pp.

Considerations of heat conduction guided the measurements made to determine the actual virgin temperatures at the "temperature stations." Holes in which were inserted one or more thermometers, were drilled a few inches back from the breast. Mercury-in-glass thermometers were chosen, mainly because of their simplicity and reliability. Two, and sometimes three, thermometers were inserted in these holes, and were read at 2-hour intervals until three or four readings had been taken. This proceeding was repeated over a number of days. It was found that reliable readings were obtained without waiting until some days after drilling and in spite of nearby blasting.

The conclusions drawn by the authors read as follows: Temperature measurements in eight special drill holes in the Calumet and Hecla mines, together with one in an old hole which has suffered no appreciable temperature change in 10 years, all fall very nearly on a straight line. The temperatures range from 74.75° F. at 3,562 feet below the surface to 95.31° at 5,679 feet. When taken in connection with Lane's value of 43° for the mean surface temperature, these give an average temperature gradient of 1° F. in 108.5 feet, or 1° C. in 59.5 meters, which is only about one-half the Kelvin average for the whole earth (1° in 27.76 meters). The gradient at an average depth of 4,500 feet is 1° F. in 103.1 feet.

The data are not sufficient as yet to allow any positive conclusions as to the time and extent of the glacial epochs, but point strongly to a value at least as large as 30,000 years as the time which has elapsed since the last epoch.--W. Ayvazoglou.

(1006) MATHEMATICAL THEORY OF HEAT FLOW IN THE EARTH'S CRUST

By David Otto Ehrenburg

University of Colorado Bulletin, Boulder, vol. 32, No. 12, May, 1932.
The University of Colorado Studies, vol. 19, No. 3, 1932, pp. 327-355.

In the introduction to the mathematical discussion of the theory of heat flow in the earth's crust given in this article, the author refers to Ingersoll and Zobel's application to physical geology of the mathematical theory of heat conduction in homogeneous media, and calculation by them of a number of curves for the cooling of lavas and laccoliths. Ehrenburg continues:

> Unfortunately, their results are based on the narrowing assumption that the thermal constants of the cooling mass are identical with the constants of the underlying or surrounding rock. However, the shape of a cooling curve depends not only upon the configuration of the bodies between which the heat transfer occurs and their initial temperature difference, but also upon the relative values of conductivity, density, and specific heat. The writer believes that the difference in thermal constants is an important factor in modifying the character of a temperature curve; and in this paper is proposed a new method of treating the flow of heat in two heterogeneous solids in contact, and also is demonstrated its application to several particular cases similar to those treated by Ingersoll and Zobel.--W. Ayvazoglou.

(1007) TEMPERATURE OF ORE FORMATION

Editorial note

The Mining Magazine, London, vol. 47, No. 5, 1932, pp. 315-317.

This is an abstract of H. C. Boydell's article published in the Bulletin of the Institution of Mining and Metallurgy for April and May, 1932, Nos. 331 and 332, under the title, "Temperature of an epithermal ore deposit," the example investigated being the Camp Bird Mine at Silverton, Colo. (See Geophys. Abs. 40.)--W. Ayvazoglou.

(1008) RATE OF TEMPERATURE CHANGE

Editorial note

The Oil and Gas Journal, Tulsa, vol. 31, No. 3, 1932, p. 109.

In this article temperature gradients for a series of measurements are enumerated: In certain wells in Pennsylvania the temperature gradient indicated an increase of 1° F. for each 58 to 64 feet of depth, and in California an increase of 1° for each 52 feet is often observed. Other temperature investigations show that in some districts the temperature rises at an even faster rate, often at 1° for each 40 feet.

In the Salt Creek field of Wyoming well temperatures at a depth of 100 feet ranged from 51.4 to 65.27° F., and at 2,000 feet they ran from 79° to 98.4°. Temperature measurements in a well in the Teapot Dome field, Wyoming, showed 71.6° at 1,000 feet and 125° at 2,867 feet. In another well in the same field the temperature at 25 feet was found to be 76°, but at 2,790 feet it was 124.5°. A completely dry well in the Teapot Dome field showed a temperature of 52° at 100 feet and of 101° at 2,000 feet. In wells in California the temperature at 4,000 feet varied between 150° and 170°. A thermometer run to the bottom of a well 4,500 feet deep in the Wellington-Fort Collins area of Colorado recorded 157°.. This was 3° higher than that registered by a well 7,900 feet deep near Kane, Pa.

Temperature measurements made by E. M. Hawtoff, of the Bureau of Economic Geology, in a well in the Big Lake field in Reagan County, Tex., showed the following temperatures at depths below 5,000 feet: At 5,700 feet, 122° F.; 6,500 feet, 135; 7,000 feet, 152; 8,000 feet, 161; 8,300 feet, 170.--W. Ayvazoglou.

7. <u>UNCLASSIFIED METHODS</u>

(1009) SCIENCE AND PRINCIPLES OF GEOPHYSICS

By C. A. Heiland

Engineering and Mining Journal, New York, vol. 133, No. 5, 1932, pp. 286-288.

In this article Heiland gives a review of Volumes I, II, and VI of B. Gutenberg's "Handbuch der Geophysik." Sections of the Handbuch der Geophysik appeared previously were reviewed by Heiland in the January, 1932, issue. Contents of the volumes were given in Geophysical Abstracts Nos. 12, 33, and 34.--W. Ayvazoglou.

(1010) SOME OBSERVATIONS OF THE BEHAVIOR OF EARTH CURRENTS AND THEIR CORRELATION WITH MAGNETIC DISTURBANCES AND RADIO TRANSMISSION

By Isabel S. Bemis

Proceedings of the Institute of Radio Engineers, New York, vol. 19, No. 11, 1931, pp. 1931-1947.

This paper presents correlations between the abnormal earth currents noted during magnetic storms and transoceanic radio transmission on both long and short waves. The radio transmission data were collected on the telephone circuits operating between New York and London and between New York and Buenos Aires. The earth-current data were collected on two Bell System lines extending approximately a hundred miles north and west from New York. The results of this work establish facts which have been known in a general way for some time.

The direction of flow of abnormal earth currents in the neighborhood of New York seems to be along a northwest-southeast line. Coincident with such abnormal currents are periods of poor short-wave radio transmission. However, on long waves, daylight transmission over transatlantic distances is improved. On the short-wave circuit to Buenos Aires, transmission is adversely affected but only to a moderate extent.--Author's abstract.

(1011) THE OPERATING FREQUENCY OF REGENERATIVE OSCILLATORY SYSTEMS

By Hugo Benioff

Proceedings of the Institute of Radio Engineers, New York, vol. 19, No. 7, 1931, pp. 1274-1277.

The operative frequency of regenerative oscillatory systems is quantitatively derived in terms of the natural frequency, the damping constant, and the phase of the driving force. As an example the results are used to calculate the change in rate of a pendulum clock due to a given variation in the phase of the driving impulses. Applications to other types of systems are briefly indicated.--Author's abstract.

(1012) GEOPHYSICAL SURVEYS

Editorial note

Oil News, London, vol. 31, No. 1016, 1932, p. 435.

According to this note, the first geophysical surveys of Trinidad, carried out by the United British Oil-fields of Trinidad, yielded interesting results. Those obtained from Eötvös balance measurements made in the southwest corner of the island are still being worked out in Europe and will be available shortly.

Magnetometer and seismic surveys have been undertaken of the area between the Pitch Lake and Aripero, but the results, if anything, tend to give an unfavorable impression of oil prospects in the area. Both surveys indicate a large fault with a throw of 900 to 2,100 feet running to the south of the lagoon. In order to get readings from the harder or high velocity beds in the seismic survey, heavy shots of 1,000 pounds of dynamite had to be fired 3 to 4 kilometers out on both sides of the fault line.--W. Ayvazoglou.

(1013) LA GÉOPHYSIQUE APPLIQUÉE À LA PROSPECTION

(GEOPHYSICS APPLIED TO PROSPECTING)

Editorial note

Le Génie Civil, Paris, vol. 100, No. 4, 1932, p. 102.

The use of gravimetric, magnetic, seismic, and electrical methods of prospecting is mentioned briefly.--W. Ayvazoglou.

(1014) LES MÉTHODES SÉISMIQUES ET GRAVIMÉTRIQUE DE PROSPECTION DU SOUS-SOL

(SEISMIC AND GRAVIMETRICAL METHODS OF SUBSOIL PROSPECTING)

Editorial note

Le Génie Civil, Paris, vol. 100, No. 16, 1932, p. 405.

Brief summaries are given of articles published in the Annales de Mines, vol. 20, No. 10, 1931, by A. Maillet and J. Bazerque, and in vol. 20, No. 11, 1931, by M. H. Galbrun. Abstracts of these articles were published in Geophysical Abstracts No. 35, p. 387, and No. 37, p. 441, respectively.--W. Ayvazoglou.

(1015) USE OF GEOPHYSICS IN PROSPECTING FOR OIL AND GAS POSSIBILITIES NOT MYSTERIOUS

By K. C. Heald

The Oil and Gas Journal, Tulsa, vol. 31, No. 4, 1932, pp. 41-43.

The author says that although there is much that is difficult to learn, and a great deal that remains to be learned about geophysical methods, there is nothing mysterious. Of course, a clear understanding of the fundamental rules is necessary for the application of geophysics.

Explanations of some principles of geophysics, especially concerning oil prospecting, are given under the following headings:

Anticlinal theory. The theory developed now is that oil and gas tend to move through porous formations from regions of higher toward regions of lower pressure, and where this movement is arrested oil and gas pools form.

Search for structure. There are three primary requisites for an oil pool: (1) Favorable structure, including lenticular sand and fissured limestone conditions that lead to oil accumulation, (2) a suitable reservoir rock of "pay," and (3) an adequate source for the oil. It is to this search for structure that geophysical methods of prospecting are applied. The method most widely used is the gravimetric and the instrument is the torsion balance.

Pull of gravity. The torsion balance will reflect changes in structure of the rocks, and will accurately reflect such phenomena as anticlines, synclines, and large faults. However, many factors render the interpretation of torsion-balance data into terms of oil-field structure difficult and, at times, impossible. Every hill, every heavy bowlder close beneath the surface, will interfere with the recognition of the effects due to structure.

Different pulls. There are areas where the geologic structure is so complex that in spite of the existence of beds strongly contrasting in mass, the instrument is affected by so many pulls from different directions that it has proved impossible to convert the instrumental records into a reasonable

geological picture. Thus the conclusion is drawn that the torsion balance will work most effectively in areas where the surface is flat, where there are no hidden "pockets" of either light or heavy material hidden close below the surface, where there are certain units among the formations within drilling depth that are appreciably heavier than those overlying them, and where structural conditions are comparatively simple. The torsion balance is totally unsuitable for work in areas where the surface is cut into high ridges and deep valleys, where the rocks at a depth of 1,000 feet or more are closely folded or complexly broken by faults, or where the rocks down to the limits of drilling depth furnish no appreciable contrast in the mass of individual units of formations.

Cost considered. Exclusive of the original cost of the equipment a torsion-balance party with two balances should be operated for approximately $3,000 a month. During that time they should make approximately 120 determinations of gravity. The territory they will cover, and therefore the cost per acre, will depend upon the detail which is essential in view of the expected geological conditions. According to data obtained from the Mid-Continent area a party can cover about 60 square miles at a cost of between 8 and 9 cents per acre. Very detailed examination may increase the cost to about 30 cents per acre.

Seismic methods. Physical principles of seismic methods are given briefly, and their subdivision into "refraction" and "reflection" methods is explained.

Search for salt domes. The refraction method has been particularly effective in discovering salt domes which lie less than 5,000 feet below the surface. The refraction seismograph is particularly suitable to explore territory where there are neither very elastic beds, such as heavy limestone, nor incoherent beds; where the formations down to a depth of at least several hundred feet are soft shales and sandstones with a highly elastic bed, and where the geologic structure is relatively simple. It is not recommended for work in areas where rocks are sharply folded, with individual beds inclining more than 30° from the horizontal, or where there are closely spaced faults.

Refraction seismograph cost. Exclusive of the original cost of the equipment an effective refraction seismograph party will call for an expenditure of from $16,000 to $22,000 per month. In that time they should explode about 150 charges of dynamite for depth determinations. The cost per acre varies greatly. Reconnaissance surveys for the discovery of salt domes have been made for as little as 4 cents an acre, while detailed surveys under difficult conditions may cost more than $5 an acre.

Reflection method. This method has been proved effective in some areas where refraction work is unsatisfactory, either because of excessive cost or because of geological conditions.

Computing vibration speed. If the seismograph operator knows that a certain per cent of the rocks down to a depth of, say, 6,000 feet is shale,

another per cent is heavy beds of limestone, and the remainder are sandstones, the geophysicist will compute a reasonable speed, based upon experience in other areas and upon the known average velocities of seismic waves in shales, limestones, and sandstones.

Most favorable conditions. Favorable conditions, as well as those which are unfavorable for application of geophysical methods, are discussed.

Reflection seismograph cost. The over-all cost per troop ranges from $8,000 to $12,000 per month. An average of 4 to 8 depth determinations can be made per day, depending upon the conditions under which the work is done and the precision attempted. Under conditions such as exist in the Mid-Continent region, the cost per acre will range from as much as 90 cents to as little as 5 cents.

Magnetic method of geophysical prospecting is mentioned, but although unquestionably more territory in the United States has been covered by the magnetic than by any other geophysical method, it has fewer oil-field discoveries to its credit than has either the gravimetric or the seismic work. Besides, the difficulty of interpretation makes this method less applicable than the two other methods.

Cost considered. For a cost of not more than $400 it should be possible to cover approximately 200,000 acres a month by magnetic methods commonly used, making the cost of this examination approximately $0.002 per acre.

In conclusion, the author notes that none of the methods mentioned attempt to discover oil or gas directly. Their entire purpose is to secure geological information.--W. Ayvazoglou.

(1016) GEOPHYSICAL METHODS LOCATE METEORITE

By J. J. Jakosky

Engineering and Mining Journal, New York, vol. 133, No. 7, 1932, pp. 392-393.

Results of geophysical investigations of the Meteor Crater were given in a previous article, Geophysical Examination of Meteor Crater (see Geophys. Abs. 25, p. 127).

From these results exploration was recommended in five locations. According to the present article two of these holes have been completed, both of them definitely checking the indications of the geophysical survey as regards the occurrence, location, and depth of the meteoric material, as well as the depth to water and other structural effects.--W. Ayvazoglou.

8. GEOLOGY

(1017) GEOLOGIC INTERPRETATIONS FROM ROTARY WELL CUTTINGS

By Robert M. Whiteside

Bulletin of the American Association of Petroleum Geologists,
vol. 16, No. 7, 1932, pp. 653-674.

The paper includes the history, improvement, and possibilities of geologic interpretation of strata in the Mid-Continent area from rotary-drill cuttings by the use of microscopy. The field technique with its certain advances in obtaining drill cuttings forms Part I. The laboratory technique of preparation, examination, and correlation of rotary-drill cuttings forms Part II.--Author's abstract.

(1018) BOGGY CREEK SALT DOME, ANDERSON AND CHEROKEE COUNTIES, TEXAS

By H. J. McLellan, E. A. Wendlandt, and E. A. Murchison

Bulletin of the American Association of Petroleum Geologists, Tulsa,
vol. 16, No. 6, 1932, pp. 584-600.

Boggy Creek salt dome is located near the axis of the east Texas geosyncline in Anderson and Cherokee Counties. It is of interest principally because it is the only interior salt dome in east Texas or Louisiana on which an oil field has been developed. Surface geology first attracted attention to this area, and exploration was carried on later by core tests and geophysical work.

This dome has many of the physiographic and structural features characteristic of other interior domes, but is unusual in shape, size, and other respects. A structurally low central area is found on top of the dome. Faulting is present near the south end of the uplift.

The salt movement probably occurred contemporaneously with that of other east Texas domes, being most pronounced between middle Wilcox and Carrizo times. In the central and northern portions of the dome, salt movement also occurred after the deposition of the Lower Claiborne beds.

Oil and gas is produced from the Woodbone formation in a long, narrow area on the southeast flank of the dome. The oil-producing area contains about 200 acres and the gas area about 50 acres. Present average daily production is about 1,000 barrels. Ultimate recovery from this field is not expected to exceed 4,000,000 barrels.--Authors' abstract.

(1019) FAULTS IN COMODORO RIVADAVIA OIL FIELD, ARGENTINA

By Enrico Fossa-Mancini

Bulletin of the American Association of Petroleum Geologists, Tulsa, vol. 16, No. 6, 1932, pp. 556-577.

This article is a summary of the material published on the structural conditions of the Comodoro Rivadavia oil field by the Bureau of Mines and the oil administration of the Department of Agriculture of the Argentine Republic. The formations are much faulted, with resulting complications in structure. Other difficulties are irregular bedding, pinching sand bodies, and variable cementation. It is difficult in this area to map underground structure from surface indications.--Author's abstract.

(1020) ON THE MECHANICS OF MOUNTAINS

By Harold Jeffreys

Geological Magazine, London, vol. 68, No. 808, 1931, pp. 435-442.

It appears that the geophysical approach to the problems of crustal dynamics leads to a theory of mountain formation that covers the main features of the formation of nappes and is in general agreement with other geological evidence. The chief novelty is in the recognition of the importance of the spreading under their own weight of the rocks uplifted as a result of crustal shortening.--Author's abstract.

9. NEW BOOKS

(1021) Johannsen, Albert. A descriptive petrography of the igneous rocks. Vol. 1, University of Chicago Press, 1931, 267 pp., 145 figs. Price $4.50. Introduction, textures, classification, glossary.

(1022) National Oil Scouts Association of America (Inc.). Year Book, 1932. May, 1932, 273 pp., 12 maps, statistical tables. Price $5.00. Obtainable through J. W. Selby, Shell Petroleum Corporation, Dallas, Texas. This yearbook is a compendium of the oil, gas, sulphur, and potash operations and the development of the south and southwest.

(1023) National Research Council. Transactions of the American Geophysical Union, Thirteenth annual meeting, April 28 and 29, 1932, Washington, D. C. National Academy of Sciences, Washington, D. C., June, 1932, 401 pp. The reports and papers presented at this meeting are given according to the sections into which the Union is divided. The contents of the book are as follows:

General assembly (pp. 5-49). - (1) Resolutions adopted: On the urgent need of continuing oceanographic work without material curtailment; on gravity work at sea by the United States Navy; regarding the purchase of apparatus for determining gravity at sea; on Naval Observatory time-signals; on the need of nongovernmental research institutions in meteorology; in commendation of Geophysical Abstracts, published as an information circular of the U. S. Bureau of Mines; on the Second International Polar Year, 1932-33; on the death of Alfred Judson Henry; on the depth of Robert DeCourcy Ward; on the death of Louis Bauer. (2) symposium on the application of geophysics to ocean basins and margins: Introduction, by Richard M. Field; Problems of island-arcs and ocean deeps, by Walter H. Bucher; The structure of the Bartlett Trough, by Stephan Taber; Seismic zones as related to relif of ocean-bottom, by N. H. Heck; Interpretation of gravity-anomalies and sounding-profiles obtained in the West Indies by the international expedition to the West Indies in 1932, by Harry Hammond Hess; Sounding the depths of the ocean for mapping the conformation and topography of the bottom, by G. W. Littlehales; The applications of seismic methods to submarine geology, by E. DeGolyer; Torsion-balance surveys in southwest Louisiana and southeast Texas by D. C. Barton; Experiences of a seismologist with "seismic methods," by A. L. Day; Discussion following symposium.

Section of Geodesy (pp. 51-88). - (1) Progress report on the absolute determination of gravity at Washington, by Paul R. Heyl; (2) Improvements in the gravity apparatus of the United States Coast and Geodetic Survey, by E. J. Brown; (3) Gravity observations in the Bahamas, by J. P. Lustene; (4) Isostasy and related subjects, by F. A. Vening Meinesz; (5) Progress-report on radio dissemination of the national primary standard of frequency, by J. H. Dellinger; (6) Geodetic control for the plotting of the aerial photographs of the Belcher Islands, by Noel J. Ogilvie; (7) Analytical methods in aerial photogrammetry,

by Earl Church; (8) New comparisons of the invar wires of Mexico, by Manuel Medina; (9) Progress report on investigation of invar tapes and precision circles, by L. V. Judson; (10) Geodetic work accomplished from May 1, 1931, to April 30, 1932, by William Bowie; (11) Further studies on the lunar correlations with small changes in the variation of latitude, by Harlan T. Stetson.

Section of seismology (pp. 89-110). - Symposium on the application of seismology to the study of ocean-basins: (1) Seismology and the ocean-basins, by N. H. Heck; (2) Accuracy of epicenter determinations, by Frank Neumann; (3) Applications of interferometric tiltmeters in the problems of geophysics, by George E. Merritt; (4) Seismology and structural geology, by W. T. Thom, jr.; (5) Earthquakes in the north Atlantic as related to submarine cables, by V. P. de Smitt.

Section of Meteorology (pp. 111-142). - (1) Weather Bureau program for the Second International Polar Year, by W. R. Gregg; (2) The use of polar year data in the study of atmospheric interchange, by W. J. Humphreys; (3) A program for the observation of weather changes during the total solar eclipse of August 31, 1932, by S. P. Fergusson and Charles F. Brooks; (4) Determination of atmospheric turbidity, by Herbert H. Kimball; (5) On winds in the upper atmosphere, by E. O. Hulburt; (6) Winds of the Antarctic, by W. C. Haines; (7) a study of the interrelation between air-temperatures in California and ocean temperatures in the northeast Pacific and its influence on long-range forecasting, by A. F. Gorton; (8) Notes on the exchange of energy between ocean and atmosphere, by W. F. McDonald; (9) Weather-charts for the northern hemisphere, by E. B. Calvert; (10) Fifty years of North American rainfall, by Oliver F. Fassig; (11) Atmospheric water-vapor, by Frederick E. Fowle.

Section of Terrestrial magnetism and electricity (pp. 143-191). - (A) Reports on magnetic and electric work of organizations in the United States during 1931-32: (1) Summary of reports received, by Harlan W. Fisk; (2) Auroral station at the Alaska Agricultural College and School of Mines, by Veryl R. Fuller; (3) Work of the Bell System relating to terrestrial magnetism and electricity during 1931, by Otto B. Blackwell; (4) Magnetic investigations of the Carnegie Institution of Washington, May 1931 to April 1932, by J. A. Fleming; (5) Researches at Mount Wilson Observatory of the Carnegie Institution of Washington relating to terrestrial magnetism, by Seth B. Nicholson; (6) Colorado School of Mines, by C. A. Heiland; (7) Massachusetts Institute of Technology: (a) Cooperation in the cosmic-ray survey, by Ralph D. Bennett; (b) The elimination of night-course variations in radio range-beacons, by N. G. Kear; (c) Developments in geophysical prospecting by electrical-potential methods, by W. Spencer Hutchinson. (8) Solar-constant work by the Smithsonian Institution, by C. G. Abbot; (9) Research in atmospheric electricity at the Stanford University, by Joseph G. Brown; (10) Geophysical work at the United States Bureau of Mines, by Scott Turner; (11) Magnetic work of the U. S. Coast and Geodetic Survey, by R. S.

Patton; (12) Work of the U. S. Hydrographic Office in terrestrial magnetism and electricity, by W. R. Gherardi; (13) Work related to terrestrial magnetism and electricity of the Naval Research Laboratory, by E. D. Almy; (14) United States Signal Corps, by G. E. Kumpe. (B) Papers: (1) On calculations of the ionization in the upper atmosphere, by E. O. Hulburt; (2) The geophysical significance of radio measurements of the ionized layer, by M. A. Tuve; (3) radio exploration of ionization of the upper atmosphere, by J. H. Dellinger; (4) Kennelly-Heaviside-layer measurements on the Byrd Antarctic Expedition, 1929-30, by Malcolm P. Hanson; (5) Some common periodicities in radio transmission-phenomena, by G. W. Kenrick and G. W. Pickard; (6) Progress in the studies of cosmic correlations with radio reception at the Perkins Observatory, by Harlan T. Stetson; (7) Slow-moving ions in the atmosphere, by G. R. Wait and O. W. Torreson; (8) Principles of a new portable electrometer, by Ross Gun; (9) Optically compensated variometers and wide-range recorders to be used during the Jubilee Polar Year, by H. F. Johnston; (10) The relation of lightning discharges to changes in the electrical field of thunder storms, by J. C. Jensen.

Section of oceanography (pp. 193-263). - (1) Progress in the Woods Hole Oceanographic Institution and cruises of the Atlantic, by H. B. Bigelow and C. Iselin; (2) Oceanographic work at the Scripps Institution of Oceanography, University of California, La Jolla, California, July 1, 1932, to April 18, 1931, by T. Wayland Vaughan; (3) Contributions of the United States Hydrographic Office to oceanography, by A. B. McManus; (4) Oceanographic work of the Coast and Geodetic Survey during the past year, by Frank S. Borden; (5) On the penetration of daylight into the sea, by E. A. Hulburt; (6) Photo-electric measurements of the penetration of light in sea water, and the results of laboratory photo-electric measurements of the absorption-coefficient of sea water, by Burt Richardson; (7) Landslide modification of submarine valleys, by Francis P. Shepard; (8) Microbiology of the marine limestones, by Richard M. Field; (9) Radium content of ocean-bottom sediments, by C. S. Piggot; (10) Temperature gradients in ocean waters, by E. E. Stephenson; (11) Progress in the investigation of surface temperatures of the western north Atlantic, by Phil E. Church; (12) The recommendation of a program of hydrographical observations leading to the quantitative determination of Arctic and Atlantic interchanges, by E. H. Smith; (13) Oceanographic work of the United States Coast Guard during 1931 and plans for 1932, by N. G. Ricketts; (14) A recent tour of some oceanographic centers of northwest Europe, by Charles F. Brooks; (15) Sea temperatures by bucket on express liners, by Charles F. Brooks.

Section of volcanology (pp. 265-273). - (1) Central American volcanoes in 1932, by E. G. Zies; (2) The volatility of silica with steam, by George W. Morey; (3) Volcanologic developments in 1931-32, by T. A. Jaggar.

Section of hydrology (pp. 275-401). - (A) Annual reports of permanent committees for 1931-32. (B) Papers: (1) Recent investigations of Thiem's method for determining permeability of water-bearing materials, by L. K. Wenzel; (2) Equation for lines of flow in vicinity of discharging

artesian well, by Charles V. Theis; (3) Characteristics of run-off of southeastern Alaska, by Fred F. Henshaw; (4) Certain flood-flow phenomena of Iowa rivers, by Floyd A. Nagler; (5) Forces acting on soil moisture in relation to other fundamental functions, by N. E. Edlefsen; (6) The disposal of moisture from the aerated portion of soils, by F. J. Veihmeyer; (7) The relation of hydrographs of run-off to size and character of drainage-basins, by LeRoy K. Sherman; (8) Velocity of flow in natural streams, by H. K. Barrows; (9) Drainage-basin characteristics, by Robert E. Horton; (10) The 1929 floods on eastern North Carolina streams, by Thorndike Saville; (11) Investigations of the fluctuations of water levels in observation wells in Virginia, by R. C. Cady; (12) Investigations of the fluctuations of the ground-water table in Pennsylvania, by S. W. Lohman; (13) Hydraulic and sedimentary characteristics of rivers, by L. G. Straub; (14) New thoughts on river regulation, by Gerard H. Matthes; (15) Model research in the river hydraulic laboratory of the Massachusetts Institute of Technology, by J. B. Drisko; (16) Present status of the National Hydraulic Laboratory, by K. Hilding Beij; (17) New formulae for predicting annual runoff of some California watersheds, by A. F. Gorton; (18) Rainfall and run-off characteristics of the Delaware River Basin, by Clarence S. Jarvis; (19) Checks on the model law for hydraulic structures, by Morrough P. O'Brien; (20) A method for computing the amount of evaporation of sea water from a tank by changes in salinity, with experimental results obtained on board the SCRIPPS, August 13-24, 1928, by Burt Richardson; (21) Problems of underground water flow in the oil industry, by Morris Muskat.

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December, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

THE INTERNATIONAL CONFERENCE ON MINE SAFETY RESEARCH AT BUXTON, ENGLAND, JULY, 1931[1]

By G. S. Rice[2]

INTRODUCTION

To those concerned in attaining greater safety to life and incidentally to property in coal mining, the meeting of representatives of the national mine safety research organizations of the principal coal-producing countries, which was called by the Safety in Mines Research Board of Great Britain at its experimental station near Buxton, should have deep significance. It is notable, first, because it meant the friendly cooperation of countries not many years ago engaged on either side in the great World War; and second, because of the remarkable degree of agreement in the essential fundamental factors and methods of attaining greater safety in coal mines when it is considered that the questions were approached by different paths and that the applications dealt with widely differing natural conditions and national points of view.

Varying views, of course, were held as to the relative importance of certain factors, but the differences were of detail rather than of principle. When the meeting adjourned it was unanimously agreed that such meetings should be continued annually and that the greatest benefit was to be derived by close cooperation in the exchange of data.

Belgium, France, Germany, Great Britain, and the United States were represented by one or several members of the respective national research organization. It is of interest to note that this research work in Belgium, France, and Germany is supported directly by national organizations of mine operators or owners but placed under a scientist appointed by the respective Government, thus giving the work official standing. In Great Britain the work is carried on by a Government board, consisting of persons connected with the coal-mining industry or in mining research, appointed by the Secretary for Mines, to direct the work of research of the Mines Department. It does not, however, come under its inspection service. Its work is supported by a penny-a-ton tax on output of coal. In the United States the mine-safety research work is directly supported by the Federal Government.

In the four countries other than the United States, the findings of the research organizations may be applied in the mines but only after approval of the respective Government

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6670."

A short abstract of the original draft was given at the meeting of the coal division of the American Institute of Mining and Metallurgical Engineers, October 9 and 10, 1931, at Bluefield, W. Va. The paper includes a summary of formal addresses presented at the conference, abstracts of the discussions, and pertinent information bearing on the subjects discussed. Since this paper was first prepared the complete papers presented at Buxton have been translated from the French and German, respectively with edited discussion and have been recently published by the Safety in Mines Research Board as Paper 74, entitled "International Conference of Safety in Mines at Buxton, 1931," 1932, 67 pp. The British translations, addresses, and discussions in the foregoing have been freely drawn upon in checking the original manuscript.

2 - George S. Rice, chief mining engineer, U. S. Bureau of Mines, detailed to represent the bureau at the Buxton conference.

mine-inspection service, or ministry in charge of mining, but in the United States, as is well known, the findings of the Bureau of Mines can not be made mandatory by the Federal Government and may or may not be made compulsory by any one of the various State coal-mine inspection departments. The utilization of the bureau's recommendations is largely a matter of voluntary adoption by mining companies of those suggestions it favorably regards. It need hardly be pointed out what an immense advantage for safety it is when the mining regulations are the same throughout a country and are thoroughly enforced. As an example of what may be done: In Great Britain, since the safety research work was begun, first carried on by the mine owners themselves in 1908, enforcement of better regulations was demanded by the Mining Department, and as a result the accident rate has been reduced from 1.33 men killed annually in coal mining to 1.07 in 1930 per 1,000 men employed underground.

PITTSBURGH INTERNATIONAL CONFERENCE IN 1912

It is of interest to recall that the first international research station conference[3] was held in Pittsburgh in 1912. It was called by the Bureau of Mines four years after the original Federal experiment station had been established there (1908) and shortly after its Experimental mine near Pittsburgh had been developed (1911) for testing. That conference, attended by delegates from the countries carrying on coal-mining research, was a splendid success, and another was to have been held in Belgium in August, 1914.

COOPERATION BETWEEN GREAT BRITAIN AND UNITED STATES IN 1923

Following the World War, research in the interest of safety in coal mining was gradually resumed in the various countries, and in 1923 negotiations were conducted between the British Safety in Mines Research Board and the United States Bureau of Mines which led to an official cooperation in mine safety research. This was consummated in the following year. The cooperation called for the interchange of research workers, of materials, of instruments for standardized testing, and of progress reports in cooperative problems.

The results have been singularly successful in establishing standards of comparison and have been stimulative tc the respective staffs through the interchange of personnel. Valuable research has been carried on and publications issued thereon, also unnecessary duplication of work has been avoided. One side may be so equipped through having specialists on its staff as to be able to carry on some special work of value to both sides. Further confidence in a specific finding is given when the same result is obtained by both sides who may employ different ways of approaching the problem.

The Safety in Mines Research Board undertook, two years ago, a similar though more limited cooperation with the French experimental station at Montlucon. The favorable results of these two international cooperations led the board to call the international conference held July 11 to 16, 1931, at Buxton, England.

THE BUXTON (ENGLAND) INTERNATIONAL CONFERENCE

The delegates to the conference were greeted by Dr. R. V. Wheeler, director of research of the Safety in Mines Research Board, who had organized the conference.

The formal meeting was opened by Sir Edward Troup, K.C.B., chairman of the board, on July 11, who gave an official welcome and outlined the program. He said that while the program chiefly related to research in explosives, it would be desirable that other subjects be

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3 - Rice, George S. (compiled) International Conference of Mine-Experiment Stations; Bull. 82, Bureau of Mines, 1914.

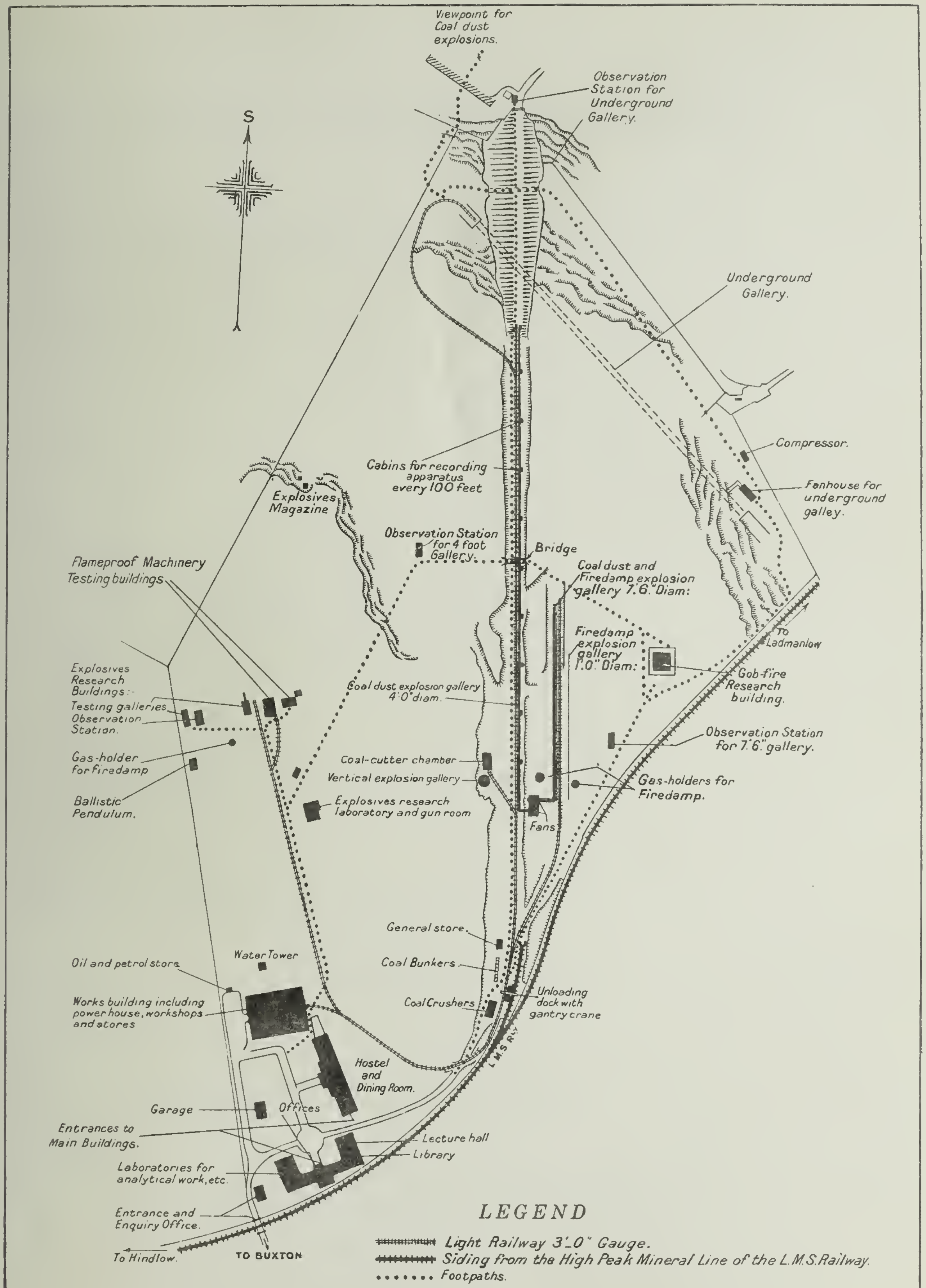
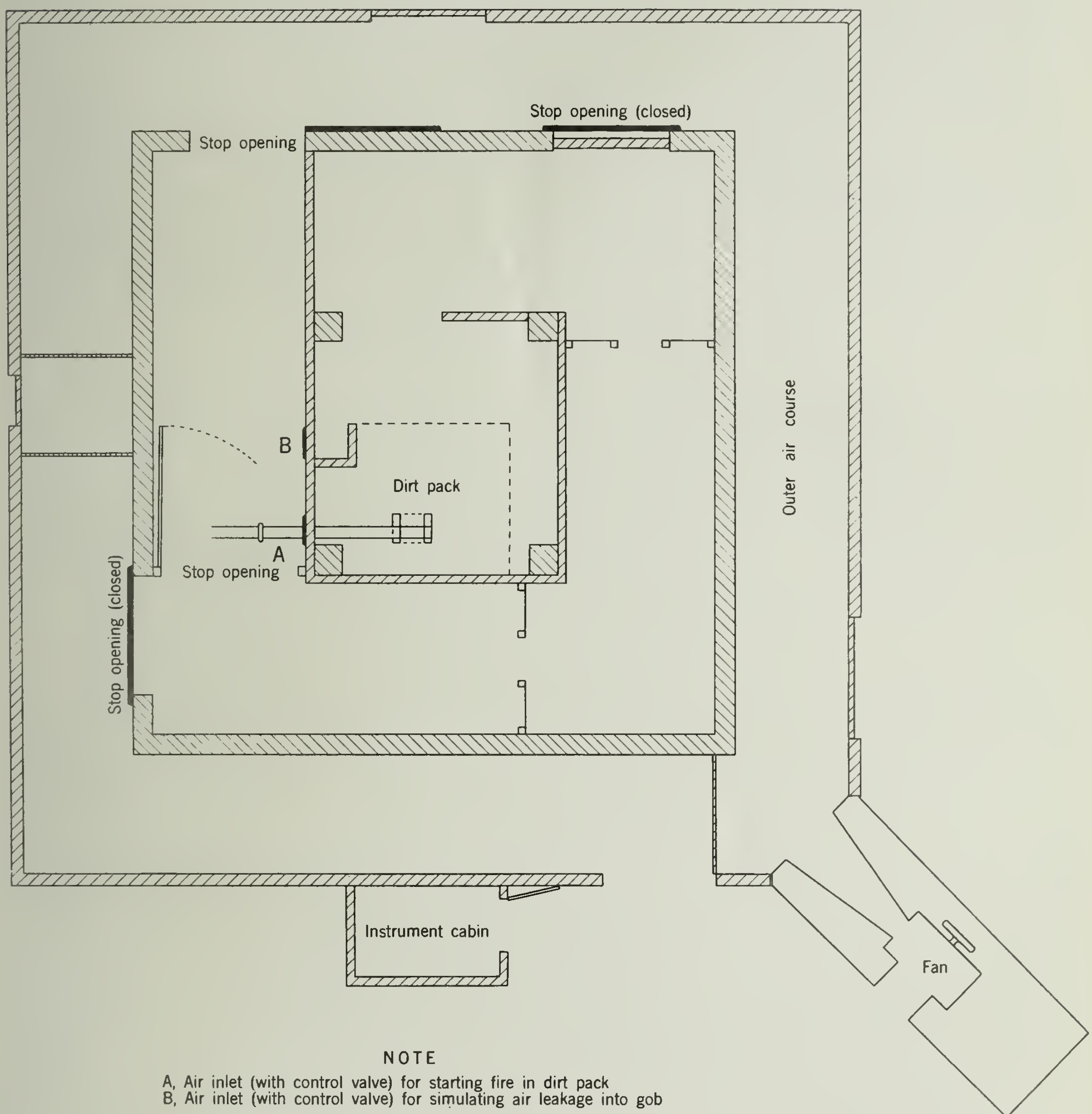


Figure 1.— Plan of Safety in Mines Research Station, Buxton



NOTE

A, Air inlet (with control valve) for starting fire in dirt pack
B, Air inlet (with control valve) for simulating air leakage into gob

Figure 2.—Plan of gob-fire chamber as arranged for experiment 14

discussed, such as the essential safety requirements for flameproof enclosures for electrical mining apparatus, and accidents from falls of roof. He briefly referred to the existing co-operations in mine safety research as between Great Britain and the United States, Great Britain and France, and France with Belgium. In concluding he hoped that the delegates to the conference would see their way to formulating a proposal of continued interchange of information on safety research.

Replies made by Mr. Audibert of France, Dr. Beyling of Germany, Mr. Breyre of Belgium, and by the writer, expressed their pleasure at attending the conference and their personal desire to aid in the promotion of the proposed wider cooperation in research. After remarks by Wheeler on the details of the conference program a preliminary tour of the splendidly equipped station was made. The layout of the station is shown in Figure 1.

Weekly Demonstrations for Visiting Miners

On the following day, Sunday, the delegates were invited to attend one of the instructive "demonstrations" held each Sunday during the summer months for parties of miners and mining officials coming from one or another of the collieries or mining centers within reach by bus travel. Usually this means coming from the nearby Yorkshire coal field, but occasionally the men are brought by train from distant fields. The parties consist of from 50 to 100 men but sometimes are larger. On this occasion (July 12) a party of about 100 came on special busses from a Nottingham group of collieries.

Dust-explosion Demonstration in Underground Gallery

In accordance with the usual program for such visits, the miners were given a demonstration of a coal-dust explosion in an underground gallery or tunnel driven through a spur of a plateau. During a test it is closed at one end by a heavy steel portcullis door. The tunnel is about 800 feet in length, 8 feet wide, and 8 feet high, and is connected near the door end by a side drift to a fan. At present this underground gallery is not used for precise coal-dust and gas explosibility tests; these are conducted in an outside steel gallery 4 feet in diameter and 1,000 feet long, equipped with United States Bureau of Mines manometers. The tunnel is constructed through a spur of a high plateau on which the extensive research plant is placed. The tunnel has been driven through a limestone stratum which is much creviced and shattered. It is lined nearly all the way by either concrete or steel arching, but the lining is not tight and the tunnel is so very wet that it might be expected that coal-dust explosions could not be produced; however, just as in the Bruceton experimental mine, when in summer the humidity of the ventilating current normally is high and the walls and floor become wet (that is, if a drying system is not used), there is no difficulty in causing strong coal-dust explosions by coal-dust newly strewn along the passageway on its ribs, timbers, and floor.

Another part of the Sunday demonstration program at Buxton is to divide up the parties into groups of 15 or 20 persons, place each under a member of the staff, who conducts his group to each of the galleries and test chambers and explains its purpose.

Gob-Fire Testing Chamber

A test chamber that attracts general attention of mining men is one designed for the study of gob fires and their control. A plan of the chamber is shown in Figure 2. It has been definitely shown repeatedly by controlled tests that a gob fire will generate explosive gases, hydrogen and hydrocarbons, that under certain test-controlled conditions of admission

of air, simulating leakage through stoppings or walls, the gas-air mixture will ignite at the fire and cause severe explosions without the prior introduction of methane. Reports on this research and many others have been published by the Safety in Mines Research Board either as progress or final reports.

Ignition of Fire Damp by Frictional Sparks

An impressive test, which illustrates the necessity of frequent examination for gas while machine cutting, was demonstrated in a gallery by the ignition of methane in operating a mining-machine while its bits were cutting into pyritic balls or, in other tests, hard quartzitic sandstone of the kind which underlies some coal beds. Another test demonstrating the ignition of an air-methane mixture by frictional sparks was made in a box by striking with a hand-pick on a particular type of hard, sandy rock.

Possible Ignition of Fire Damp by Compression from Shot

The ignition of an air-methane mixture by adiabatic compression, such as might be caused by a blast in which the explosive gases are projected through a crevice to an opening containing a methane-air mixture is shown experimentally by suddenly compressing a methane-air mixture in a cylinder which automatically raises the temperature of the mixture sufficiently to ignite it; then the flame produced passes through a small hole in the cylinder head to another chamber containing a similar inflammable mixture, and ignites that.

Many other researches are carried on of a fundamental character; in fact, the British board is carrying on at the present time more kinds of fundamental research in coal-mine safety work than is being done in any other countries.

Lectures to Miners and Discussions

The miners' Sunday program includes taking up some practical mining question in a lecture hall. After a talk by a member of the staff, discussion follows in which the miners, as well as their officials, are urged to take part. This is not only educational to the miners but is of value to the staff of the board in obtaining the miners' point of view. The subject discussed, led by Major H. M. Hudspeth, at the meeting attended by members of the international conference, was on falls of roof and roof control by systematic timbering.

Roof Movement and Control Studies of the Board

The staff of the board is now studying the question of support of workings in different districts of the country, and, in the Durham, Northumberland, and Scottish fields, in conjunction therewith, testing is being carried on with dynamometer props and continuous recorders of subsidence of roof and rise of floor in the working places with the object of determining the best method of support in longwall faces and also in headings of bord and pillar workings. The dynamometer props being experimented with are of two kinds: The Wazau of German design employs a cylinder containing mercury, placed axially in the telescopic steel prop; under pressure the mercury is forced out through a hole of small diameter into a graduated receptacle. The relation of the amount of mercury displaced to roof pressure, is obtained by prior calibration in a hydraulic pressure testing machine. The second type, designed by Dr. A. Winstanley of the Safety in Mines Research Board staff, consists of a steel bar with enlarged ends. To avoid the effect of its rigidness on the roof stratum, it is cushioned at the top by a wood cap piece and under its foot by wood blocks. Punch marks

are made on the four quadrants, top and bottom, and strain gages of the ordinary type employed for determining compression and tension in metal structural pieces are used at regular intervals to determine the pressure produced by the roof and floor. Calibration of the props was obtained by prior testing in a hydraulic-pressure machine.[4]

The subsidence recorders are of the Safety in Mines Research Board design. They consist of a telescopic bar and cylinder with spiral spring to hold it firmly against the roof and floor. It is arranged to telescope under compression, and, through action of a lever arm, a style or lead traces a line of relative movement due to compression or convergence of roof and floor, on paper wrapped around a revolving drum actuated by clockwork, thus obtaining subsidence records with reference to definite times. Some of the recorders register data for a period of 24 hours and others for a week.[5]

These devices and methods of testing provide new ways of studying roof control and the design, strength, and intervals apart of roof supports such as pack walls, steel props, girders, and timbers. Although it is too early to give conclusive results, the writer's observation of this work in certain mines in Great Britain convinced him that these new methods of testing in mines will give information of great practical value to mining men.

PROGRAM OF THE INTERNATIONAL CONFERENCE

Although many kinds of coal-mine safety problems were discussed informally by the delegates to the conference in the course of visits to the different kinds of testing apparatus at the Buxton station and research laboratories of the board at Sheffield, the formal program during the meetings in the lecture hall at Buxton related principally to explosives of permitted or permissible type, their behavior under certain conditions, their hazard when used in gassy and dusty mines, and cognate matters.

Address by A. Breyre on Multiple Shot Firing - Classification of Mines in Belgium

A. Breyre, administrator-director of the National Institute of Mines, the research work of which is carried on near Frameries, Belgium, in an illustrated address advanced the idea that simultaneous firing of shots in a heading or working place is safer than firing one shot at a time and attempting to test for gas between shots. Belgian coal mines are divided into four groups as respects fire damp: Mines without fire damp; mines of the "first class," which are only slightly gassy; mines of the "second class," which are definitely gassy; and mines of the "third class," which are liable to instantaneous outbursts of gas. The effect of dustiness of a coal seam is superimposed on the classification of fire damp; coals containing less than 15 per cent of volatile combustible matter are considered to produce dust not dangerous, coals containing 15 to 22 per cent are considered to produce dangerous dusts, and dusts of coals with over 22 per cent are considered of very dangerous character. The presence of the dangerous dusts in a seam being worked may raise the classification of the mines on a fire damp basis to a higher or more dangerous class.

Simultaneous Shot Firing In Gassy And Outburst Mines

The Belgian regulations permit simultaneous shot firing in mines of the first class (slightly gassy) in shaft sinking, crosscutting, and development roads. In mines of the

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4 - A dynamometer prop of similar type, a seamless steel pipe had been designed for testing roof pressure in headings in the Experimental mine of the Bureau of Mines and will be tried out in the near future.

5 - A duplicate recorder has been constructed by the Bureau of Mines for similar studies of convergence of roof and floor.

second class only one shot at a time may be fired, and external stemming (rock-dust) or a cartridge sheath (Lemaire's rock-dust sheath) must be used. Further, all shot firing in coal is forbidden in very gassy mines (class 2 and 3). There are, however, two exemptions from this regulation: Namely, in these anthracite seams under class 2, the coal of which is hard and the dust regarded as nonexplosive, and in certain mines (class 3) liable to instantaneous outbursts in which "volley" firing of shots to shatter the coal in the faces of certain workings is done to induce threatening outbursts. But volley firing is permitted only under the condition of firing from the surface or from protected or refuge plans or chambers when all men are out of the mine or affected district.[6]

Breyre described tests of multiple shot firing made by his predecessor, the late Mr. Lemaire, in the National Institute's Colfontaine gallery, a heading driven into rock in which 2 to 6 shots of permitted explosive were fired simultaneously in the respective test, into a firedamp-air mixture without producing its ignition. As a result Lemaire's opinion was that simultaneous shot firing presents no more danger than single shot firing.

Breyre made a series of tests in the institute's surface gallery, firing into fire damp and into coal-dust, without causing ignitions. He cited as confirmatory evidence that in two mines subject to instantaneous outbursts, between 1922 and 1931, 4,618 shots in 335 volleys had been fired, in one mine, 10 to 27 shots in a volley, and in the other mine, 3,181 shots with 6 to 30 shots in a volley had been fired without igniting gas, although the shots induced the occurrence of a number of gas outbursts.

As Breyre stated, these shattering shots fired in volleys are not directly comparable with normal shot firing in a heading; but they are suggestive as concerns the safety of multiple shot firing.

Breyre's Conclusions About Multiple Shot Firing and Requirements for Detonators

Breyre in his conclusions says:

(a) Series connections of shots in a single circuit are strongly recommended.

(b) Electric detonators must be carefully classified in respect to their resistence in ohms and in respect to those in which the same amount of heat will be dissipated in the same time.

In Belgium regulations require the manufacturer to class together all detonators of the same resistence and to mark the packets accordingly (for example, 1.4 to 1.5 ohms). This applies to low-tension detonators having a platinum bridge.

The majority of mines have a small apparatus for testing periodically the figures given by the manufacturer. One mine liable to gas outbursts and requiring a large number of shots in a volley specifies to the manufacturer that the resistence of the detonators does not vary by more than 0.02 ohm. With carefully selected detonators, misfires are very rare.

Of the few misfires that do occur which are chargeable to defective detonators not all are due to improper resistence, but exceptionally may result from humidity of the igniting

6 - The nature of instantaneous outbursts of gas which have occurred in certain mines in Lower Silesia, Germany, where the gas is carbon dioxide, and in certain mines in British Columbia, South Wales, France (Gard Basin), and Belgium where the gas is chiefly methane is described in a symposium on the subject given before the American Institute of Mining and Metallurgical Engineers. This is presented in papers entitled: Introductory Notes on Origin of Instantaneous Outbursts of Gas in Certain Coal Mines of Europe and Western Canada, by G. S. Rice; Instantaneous Outbursts of Carbon Dioxide in Coal Mines in Lower Silesia, Germany, by P.A.C. Wilson (Neurode, Germany), and discussions by various persons; Transactions of the Coal Division of the American Institute of Mining and Metallurgical Engineers, 1931, pp. 75-136.

material surrounding the bridge wire or to the detachment of the head of collodion which generally seals the socket of the detonator.

(c) Simultaneous shot firing necessitates careful examination of the face after each volley to detect possible misfires.

(d) The shot-firing device or unit (termed exploders in Great Britain) must be powerful; underground experience has shown that they rapidly lose some of their power in service. Breyre recommends as a general rule the application of a coefficient of 0.5 to the rating given by the manufacturers as regards the maximum number of shots the apparatus can fire simultaneously. The shot-firing device should be so designed that it can pass the current into the firing circuit only when the current has reached its full strength. If the current passed in the circuit initially was low and then increased there would be a possibility that the less resistent detonators would explode prematurely.[7]

In concluding Breyre remarked that he was under the impression that Great Britain and Belgium were the only countries not yet permitting simultaneous multiple shot firing. (The United States Bureau of Mines has not as yet recommended multiple shot firing in coal mines).

Discussion of Multiple Shot Firing Position of France

Audibert said not only was simultaneous shot firing permitted in France, but it was required. He felt that as far as ignition of fire damp was concerned simultaneous shot firing was no more hazardous than firing single shots.

In connection with the considerable delay that had been noted in the blast when firing a number of shots simultaneously, Taffanel had shown that it was impossible for any difference in the detonators to account for a long lapse of time. Audibert remarked that what he had to say referred only to French explosives and suggested that the apparent lag of detonation on firing a number of shots simultaneously may be caused by deflagration of one of the shots, which might be due to a gap between cartridges In 27 years, however, French mines had only nine recorded instances of ignition of fire damp, from shot firing.

Position of Germany

Beyling said that in Germany all coal mines were considered to be gassy and all explosives that were used in the coal mines were tested for permissibility. He claimed that no explosion had occurred in German coal mines for many years which could be attributed solely to an explosive cartridge. Such explosions as had occurred in connection with their use were, for example, due to the use of aluminum-sheathed detonators. In only one such explosion was simultaneous shot firing in use and in his opinion the explosion would have occurred even if only one shot had been fired. As the result of his tests in the Hibernia experimental mine (Ruhr district) he was not inclined to the view that it was more dangerous to fire several shots than one, but rather that firing a number of shots simultaneously was safer than firing one shot at a time.

Effect of Size of Testing Gallery.- In testing work in galleries he agreed with Breyre that the safety factor of an explosive is reduced as the cross section of the gallery is made smaller but not in proportion to the reduction. In comparing Breyre's experiments in a gallery with conditions found in practice, he pointed out that the gas which may be ignited

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7 - Subsequent to the conference, Breyre informed the writer that there were multiple shot firing devices manufactured in Austria which met the requirements of current strength and limited the duration of current to 0.03 second. Testing of these and other multiple shot firing devices is being undertaken by the U. S. Bureau of Mines at its Pittsburgh station.

in front of the shot hole is not pure fire damp but a mixture of fire damp and gaseous products of detonation.

Ignition Effect of Explosive Does Not Increase Beyond a Maximum Weight of Explosion.- In testing in the Hibernia experimental mine (near Dortmund) Beyling found that a series of shots with charges of 700 grams gave no ignitions, while another series with the same explosive with 600 grams gave an ignition on the fourth test and a series with charges of 500 grams gave an ignition almost every time. As a result of these observations the charge limits of explosives in the German test gallery are determined with charges between 400 and 500 grams in weight, as being the range in which the greatest danger exists. This supports the contention that it is not a pure fire damp mixture which may be ignited by an explosive (under the conditions of test), but when large charges are used a high concentration of the gaseous products of detonation is obtained in front of the shot hole and this probably is less inflammable than a methane-air mixture. He concluded, therefore, that within limits, the greater the charge the less is the liability of ignition.

(The writer of this paper suggests that this conclusion may not be valid for a shot in the open face of a mine working, but perhaps applies only to a gallery or heading of limited cross-sectional area).

Beyling did not consider that the misfires obtained in simultaneous shot-firing were of sufficient importance to be a reason for prohibiting the practice. He suggested that Breyre's specifications of range for resistence variations of detonators was narrow. This was not the requirement in Germany and he did not think that two detonators with exactly the same resistence would operate more reliably than those of which the resistence varied between, say, 1.4 and 1.5 ohms. He could not see the practical value of laying stress on precise resistence, such as mentioned by Breyre, - a tolerance allowed manufacturers of only 0.05 ohm, - particularly as the measurements are made in a laboratory, which he (Beyling) did not consider a final criterion. In practice, the solder of the bridge wire may absorb so much heat that the detonator will not fire, and there are factors inherent in use of the detonator that must be taken into consideration. In any event the detonator should always operate reliably, irrespective of small differences in resistence.

Beyling, turning to the question of the possibility of ignition of fire damp by sparks from a shot-firing unit (exploder), said he felt that it was more important to consider the liability that the detonator leads will be pulled out of the hole and make a spark on contact. The probability of this occurring is greater with one shot than with a series of shots and becomes progressively less as the number of shots is increased.

Rother (chief of the German Mine Safety Department, Berlin) agreed with the previous speakers as to the greater safety of multiple, simultaneous shot firing over single shot firing. He said that a practical view of the whole problem must be taken, and that ignition of fire damp might depend as much on other factors as the number of shots fired - for instance, the length of the shot hole as well as the length of the stemming. He did not believe that with multiple shot firing the roof would suffer if proper care was exercised.

Position of Great Britain

Wynne of the British Mining Department stated that there was an ever-present fear among miners that cartridges from misfires might be lost in the coal. He said, however, that the experiments of Breyre had demonstrated the value and utility of simultaneous shot firing.

Discussion About Safe Exploders.- Wynne asked whether an exploder were available which was incapable of igniting fire damp.

Breyre replied that he knew of no exploder incapable of igniting fire damp, other than a dry battery.

Audibert agreed that he knew of no inherently safe and efficient exploder, but thought that a greater measure of safety could be obtained by limiting the time during which the electric circuit was closed.

Allsop of the Safety in Mines Research Board staff described the construction of an exploder which had been designed and built at the Sheffield laboratory in which all inductance had been eliminated and in which the circuit was closed for a very brief period. This exploder was inherently safe. (It had not yet been officially approved by the Mines Department.)

Payman of the Safety in Mines Research Board, when called upon by Wheeler, said that in Great Britain the regulations are practically the same as in Belgium, but that the reasons for forbidding simultaneous shot firing were different - for one thing, the difficulty of apportioning the work of a series of shots with the increased risk of a blown-out shot. Personally, he did not believe that there was any danger of ignition from a blown-out shot. Secondly there was a fear that the shots might not go off together; one might be fired before another. Further, the present British regulations take account of the possibility that one shot may do so much work as to render undesirable another shot in close proximity.

As regards misfires with this present system of single shots, about 1 misfire occurs to 2,000 shots fired. Other questions raised are the effect on roof control and the fumes formed by many shots fired simultaneously. These objections of the mining men will have to be overcome to introduce simultaneous shot firing in Great Britain.

Position of United States

The writer of this paper when called upon to discuss Breyre's address remarked how valuable the studies of Breyre and those other persons entering into the discussion would be to coal-mining men in the United States, particularly in mines where they are faced with the question of shot firing in pitching coal beds, or in gassy mines where the roof conditions are bad.

The problem of multiple shot firing is under consideration by the Mine Safety Board of the United States Bureau of Mines, which makes recommendations to the director of the bureau on questions of mine safety policies. The bureau has no mandatory power over mines, but like the Safety in Mines Research Board publishes recommendations based on tests and research. These recommendations may or may not be adopted by the different State mining departments which have coal mining to deal with. In the case of approvals of mining machines, devices, and explosives for use in gassy and dusty mines, and which present ignition hazards, the permissible specifications are usually accepted by the mining departments of the principal coal-producing States and in many instances are voluntarily adopted by leading mine operators.

<u>Establishment of United States Bureau of Mines Pittsburgh Mine Safety Testing Station in 1908 Recommendations of an International Committee</u>.- When the work of the Bureau of Mines was first started in 1908, the late Joseph A. Holmes, first director, enlisted the help of those representatives of Great Britain, Belgium, Germany, and France in charge of mine safety research. It was not possible, however, at that time to obtain the help of J. Taffanel, then director of the Lievin station, France. But a commission consisting of A. Desborough of the Explosives Department of the Home Office of Great Britain, Victor Watteyne, Inspector General of Mines, Belgium, and Carl Meissner, Mining Councilor of Germany, visited the United States in 1908, and their recommendations were influential in formulating the methods of testing and use of explosives in gassy and dusty coal mines. This testing began at Pittsburgh in the fall of 1908.

Specifications of Permissibility of Explosives; Recommendation of Single Shot Firing.- Among the specifications for permissibility of an explosive were that its (maximum) charge limit be 1½ pounds, that the drill hole should be properly located, the charge stemmed with clay, and only one shot at a time prepared and fired after examination by safety lamp of the atmosphere in the working place had determined that no gas within the limits of detection by a flame safety lamp was present. The specifications requiring the use of detonators with electric shot firing, discharging but one shot at a time, were not added until later.

If shot firing was done electrically from the surface when all men were out of the mine, a method then and now required in coal mines in the State of Utah, the foregoing recommendation of firing one shot at a time was waived. Other recommendations were made by the international committee, but these strongly influenced the adoption of the single shot firing policy of the bureau.

Shot Firing from the Surface[8].- In mines where shot firing from the surface has been done in the States of Utah, Colorado, Oklahoma, and Alabama, there were ignitions of fire damp, but it is concluded that these were due to one of the following reasons:

(1) Use of a strong and high-voltage current up to 250 or 550 volts with the tendency to leakage at temporary connections.

(2) Defective wiring of circuits and use of uninsulated wires.

(3) Prolonged leaving on of current, or throwing in the firing switch again.

The use of current of too high a voltage seems unwise on account of the temporary nature of shot-firing lines in the workings parts of the mine. Hence in some mines employing the method, the voltage of the shot-firing lines has been reduced to 110 volts.

Defective shot-firing circuits are difficult to avoid completely in an extensive mine which is one objection to the system of firing all shots at one time from the surface.

Prolonged application of current and repeated closing of the shot-firing switch formerly were probably the most prolific sources of trouble in causing arcing of the electric leads brought in contact by the blasts. The condition has been remedied in some installations by use of a mechanical shot-firing switch which permits but one contact, of short duration.

The use of "delay-action" detonators, which have been employed in a few districts in multiple shot firing in coal mines, is very reprehensible, as the delayed shot may shoot into gas and dust produced by the first blasts. The Buxton conferees stated that they regarded their use as highly dangerous in coal mines.

The chief and great merit in firing from the surface is that if an explosion occurs no one is killed. Nevertheless the system for various reasons has not been extensively adopted; in fact, with the growth of mechanism and concentrated mining, its use has lessened except in Utah where it is required by State regulation.

Multiple Shot Firing by Shot Firers.- The system of multiple shot-firing in individual working places, using fuse fired by shot firers when all other men are out of the mine, is still extensively practiced in the mines of the Middle West but has not been much used in the Appalachian field where there is the largest production of coal. This system lessens the hazards of explosions to the larger number of men but increases the hazards to the shot firers because they usually are hurried and do not take time or have opportunity of seeing the charge or burden of individual shots. To remedy this, the shot firers in some mines charge and stem all shots as well as fire them, when all others are out of the mine, but many mining men regard this as impracticable and unnecessary if care is taken otherwise, especially if electric shot firing using permissible explosives is done by experienced certified foremen.

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8 - The comments that follow were not officially presented but were discussed informally at different times with members of the conference.

Demand for Multiple Shot Firing.- In steeply pitching workings, especially in gassy mines, there is real need of multiple shot firing; and it is extensively practiced, although not in accordance with existing Bureau of Mines recommendations which many progressive operators wish to follow; hence, the consideration at this time by the bureau of whether a system of simultaneous multiple electric shot firing can be made safe. The need of such a system also is urged by operators who employ mechanization extensively. The bureau is therefore investigating the question which appears to involve standardization of detonators, the duration and strength of the current required, the securing of a suitable multiple shot firing device that will not ignite fire damp and other practical blasting considerations.

Address by E. Audibert on the Testing of Explosives for Permissibility

Audibert described methods of testing explosives for use in gassy and dusty mines by the French and other stations, the determination of charge limits, and the discrepancies found in these under the different methods of testing employed by the various testing stations. Under the French law of 1890, the permissibility of explosives was not determined under a schedule of gallery tests in the presence of gas and (or) coal-dust as in other countries, but was based on certain requirements of composition and calculated detonation temperature. The French regulation of 1911 does not reaffirm this doctrine but states that only those explosives prescribed by the Minister of Public Works can be used in gassy mines. No specifications are given in his official decree as to what determines the official approval or nonapproval of an explosive.

While admitting that gallery tests of explosives have been useful in demonstrating the need of substituting safer explosives for black powder and dynamite in coal mining, Audibert expressed the view that gallery tests are not altogether satisfactory. He said that they reasoned in France that the charge limits of permitted explosives relate to an entirely arbitrary set of conditions, and hence, logically, a set of conditions must be fixed which give the smallest charge limit. They were successively testing at the Montlucon gallery the value of the various factors that influence the charge limits.

Based on their research he suggested that the variable factors, apart from those of chemical composition of the explosives, and character of the gallery are: The position and character of the detonator, the length and diameter of shot hole, diameter of cartridge, density of charge, the distance separating the charge from the mouth of the hole, the length and character of stemming if used, and the degree of deformation of the shot hole at the time the test is made.

> In concluding Audibert stated: The Experimental Station of the Comité Central des Houillères de France has tentatively fixed the relation which, in the absence of stemming, exists between the charge limit and the conditions of firing when the initiation is direct (detonator in the outer end of the charge) and the diameter of the cartridge is equal or nearly equal to the diameter of the shot hole. It is therefore sufficient in order completely to define his conditions of shot firing, to indicate the nature of the explosive, the value of its density, the diameter of the cartridge, the value of the weight of the charge, and the distance separating the priming point from the mouth of the shot hole.

The research carried out in this manner has led to the conclusion that, for each explosive and for each value of the diameter of the cartridge and density of loading, there is a "limiting value" for the charge which has the property, either in the presence of fire damp or of coal-dust, of not producing ignition when fired without stemming. There are,

however, for every explosive and for every charge density and diameter of cartridge, two values: The one when the shot is fired in the presence of fire damp and the other when the shot is fired in the presence of coal-dust - properties which customarily are wrongly attributed to the charge and termed "charge limit" (that is, the maximum value of the charge which does not produce ignition of fire damp and (or) coal-dust). Safety in mines would be considerably increased by standardizing experiments to determine these properties.

Discussion of Audibert's Address on Testing of Explosives

In opening the discussion, Wheeler complimented Audibert on the careful analysis that he had made of the factors influencing test conditions. He pointed out that the task of standardization of certain factors was simplified in France by the fact that only one diameter of cartridge and borehole was permitted and direct initiation by placing the detonator in the outer end of the charge was compulsory.

Beyling said Audibert had suggested valuable ideas to those testing the safety of explosives, but everyone realized the gap between conditions of test and actual conditions under which an explosive is used in a mine, and that this gap made it impossible for the gallery tests to give absolute results. No claim had ever been made in Germany that these tests were absolute, but a large number of repeated tests overcame to a great extent that disadvantage. In Germany it is the regular practice to retest explosives already in the permitted list to insure that the standard of safety is maintained. Nearly 28 years' experience in testing had led to progressive improvement. Beyling stated that since 1925 there had been five explosions in mines in Germany in which shot firing had been implicated. In three of these the exploder (shot-firing device) was to blame and in the other two the explosives were used in a manner forbidden by regulations.

Referring to the distinction between stemmed and unstemmed charges, Beyling said that 1/8 millisecond after detonation the flame and detonation phenomena have ceased to exist, a fact that makes it unreasonable to suppose that ignition of fire damp is due directly to these factors. On the other hand, the hot gaseous products of detonation persist for a long period and may play some part in the ignition. He considered it impossible that a stemmed shot could be more dangerous than an unstemmed one, and upon this view was based his policy of carrying out tests with unstemmed shots as being the more dangerous.

Audibert said in reply he did not mean to suggest that the methods of testing explosives in other countries were unreliable, but he desired to present some new aspects of an old problem.

Beyling asked if Audibert considered the French explosives not sufficiently safe when judged by his standard; and also, would German and Belgian permitted explosives be found dangerous if fired under Audibert's conditions? Audibert replied that the majority would not be found unsafe, but that his principal aim was to find safer explosives.

Address by E. Beyling on Investigation of the Igniting Power of Explosives

Beyling described the successive steps that had been used in testing the permissibility of coal-mining explosives in Germany. For many years steel mortars and galleries of various sizes had been used, and while these had been valuable in differentiating between safe and unsafe explosives, it was desirable to obtain more fundamental information as to the cause and conditions attending ignition of gas and coal-dust by explosives. Accordingly, the

experimental mine[9] at Gelsenkirchen (beginning in 1928) had been utilized to study the factors.

Previously in surface galleries at Dortmund and elsewhere Beyling and his associates had determined the length and duration of flame from explosives, its velocity, temperature, and products of detonation in connection with approving explosives for use in gassy mines. Now they were endeavoring, in the Gelsenkirchen mine, to determine what factors affect ignition. The explosive is fired in bore holes in a strong sandstone hanging wall. In certain of the tests the explosive is fired into a body of fire damp confined by diaphragms. Coal-dust had not yet been used.

One line of inquiry was to determine the size of flame from permitted explosives in blown-out shots. In this they studied the effect of the following factors: Amount of the charge, depth of the borehole, length of the free space, diameter of the cartridge, diameter of borehole, position of detonator in the charge, type of rock in which the borehole was placed, and the effect of various methods of stemming and no stemming. The principal results obtained were as follows:

(1) With a shot hole of constant depth and with the detonator in the same relative position in the charge, the flame of safety explosives decreases with an increase in weight of charge.

(2) With a charge of constant weight, the flame increases with an increase in the free space in the shot hole between the charge and mouth of the hole - that is, with increased depth of hole.

(3) With a constant free space the size of the flame is independent of the depth of the shot hole, provided that the charge is at the bottom of the hole. These results only apply to direct initiation with the detonator at the front, or to intermediate initiation, which is much favored in the Ruhr.

(4) The further the initiating charge or primer is situated within the charge - hence, the greater the number of cartridges between the detonator and the mouth of the hole - the greater is the flame produced. The largest flame was obtained with inverse initiation, when the detonator is situated at the bottom of the shot hole.

Another series of tests with safety explosives only was made in one of two chambers excavated in the rock in the lowest level of the mine, one chamber 4 square meters (43 sq. ft.) and the other 2 square meters (21.5 sq. ft.) in cross section. The difference in cross section was found to have no appreciable effect on ignition of fire damp and the narrow chamber was principally used.

The most important result obtained was that no safety explosives tested ignited fire damp when the detonator was inserted in the shot hole last - that is, with direct initiation. On the other hand fire damp was ignited when the detonator was situated in next to the last or outer cartridge (intermediate initiation) and also when it was in the first cartridge inserted in the hole (inverse initiation). This result appeared to confirm their prior supposition that the greater the size of the flame the greater the tendency to ignite fire damp;

9 - The experimental mine at Gelsenkirchen, Ruhr, Germany, was formerly the Hibernia mine. It is an old commercial mine which was started about the middle of the last century and abandoned as unprofitable in 1925. The coal beds are numerous but generally thin, and the upper more regular beds are largely worked out within the limits of the property. The lower coal beds are much crumpled and irregular. The hoisting shaft reaches a depth of about 3,000 feet. The 12 level - the lowest at a depth of about 2,920 feet - is a cross-measure drift largely in strong sandstone, and is the level in which the tests of explosives have been conducted. The mine was taken over in 1928 for experimental purposes jointly by the Ruhr Syndicate, the miner's association, and the Prussian Government; the research work is under the direction of Dr. Beyling. Some tests on explosives were observed by the writer in 1928. A series of annual reports has been made by Beyling and his associates.

for as he had previously stated in the flame experiments, they found that the flame increases with the increase of depth of the primer within the charge.

Fire damp was not invariably ignited when initiation was intermediate or inverse. In these tests contrary to the flame experiments, the free space in the shot hole, which had so marked an influence on the size of flame, had none in these tests. A shot with a small or no free span gave ignition, while the same explosive gave no ignition with a larger free space.

On comparing the results obtained with shots fired from holes of the same depth, it was found that the probability of an ignition increases as the charge is increased in spite of a reduction in the size of the flame. Yet he remarks that with the highest charges of 10 cartridges no ignitions were obtained, and with 8 cartridges nonignitions were far more numerous than ignitions. He added that this series of tests were not in accord with earlier tests and seem to indicate th.t the flame of safety explosives is not a determining factor of their relative safety.

Testing for Ignition of Fire Damp by its Compression Produced by a Blast

Beyling said that in searching for other causes of ignition they undertook to determine whether sudden compression of the fire damp in front of a blown-out shot might cause ignition. Lead plate disks were hung in front of the hole at varying distances so that the shot impinged on the center and then measured the deformation; however, the comparative measurements were so inconsistent that no satisfactory conclusion could be drawn, although there was a marked drop in deformation with distance from the mouth of the hole. At a distance of 1.50 meters (5 feet), for instance, the lead disks were so slightly deformed, even with large charges, that ignition was not likely.

Ejection of Hot Particles from Explosive

In the lead-plate tests an important observation was made that there was found after a shot a thick incrustation of undecomposed explosives on the face of the plate. Investigators in other countries, particularly Audibert, have thought ignition of fire damp might be caused by solid particles of explosive ejected from a shot hole, or to the reaction of inflammable gases from the explosive undergoing combustion outside of this hole. To determine this, shots were fired into pure oxygen gas and flame photographs were made. These showed a number of fine bright lines, which are undoubtedly due to the burning solid particles of explosive ejected from the borehole.

Confirmatory experiments to demonstrate that solid particles, either burning or in a hot state, are ejected was made by filling toy balloons with a mixture of 1/3 methane and 2/3 oxygen and suspended in front of the shot hole. These were ignited when three cartridges of Wetter-detonit B (a permissible explosive) were fired with a free space 22 centimeters (9 inches) long when placed within 2.75 meters (9 feet) from the shot hole. With five cartridges the distance was increased to 3.75 meters (12 feet). Neither the flame nor necessary compression effect extended to this distance. An additional proof that the flame was not responsible for ignition in these tests was indicated by a test in which the flame of an unauthorized (nonpermissible) explosive partly surrounded a balloon without causing its ignition, as shown by a photograph which Beyling exhibited.

To indicate that neither flame nor compression was responsible under the condition of the tests, an iron plate with seven small holes was suspended between the shot hole and the balloon; then in spite of the protective plate the balloon was ignited.

Detonation of an Explosive in a Glass Tube

To obtain some insight into the mechanism of detonation, some cartridges were inclosed in glass tubes and fired. The internal diameter of these tubes was equal to the standard diameter, 1½ inches, of the shot holes in rock.

By photographing they were able to study the development of the reaction. The detonation was so rapid that the glass tube was only destroyed after the luminous phenomena had ceased. The glass tubes were 30, 36, and 54 inches long, respectively, and were closed at one end, with a clay plug 2 inches long. The bright luminosity of the detonator fuse head is clearly seen in the photographs; a dark space appears immediately above the charge in the free space. The nonluminosity of this part of the free space was surprising, and although several theories as to the chemical reactions were given by Beyling, he comments that it is difficult to explain. Flame phenomena above the charge are obtained only under one condition - namely, when stemming tightly seals off the free space. Rock-dust in a paper tube above the explosive in the glass tube produced the same phenomena as an unstemmed shot; that is, no luminosity was above the cartridge. In contrast a cartridge was freely suspended without enclosure and fired. The photograph Beyling exhibited shows how much fainter is the luminous record than when the cartridge is enclosed in a glass tube.

To further obtain an insight into the conditions of detonation in a glass tube, a flame-speed camera with a powerful photographic lens was used which focussed on a rapidly rotating film. This had a speed of 40 meters (132 feet) per second. It was found that the detonation wave travels through the charge with a velocity of about 3,000 meters (9,900 feet) per second, while the flame in the free space moves initially at a speed of about 1,800 meters (5,900 feet) and later of about 1,400 meters (4,600 feet) per second.

Discussion of Beyling's Paper

The writer, who was asked by the chairman, R. V. Wheeler, to open the discussion, said that it was very difficult for him to contribute anything but congratulations to Beyling on his masterly paper. He had been down in the Hibernia experimental mine in 1928 and observed some of Beyling's early tests of explosives fired in holes in hard sandstone. There were several points in Beyling's address that he had not understood. For example, the determination of pressure by impact of the pressure wave on a suspended lead disk. He wondered whether it was safe to reason from its degree of distortion the relative strength of the impact at different distances from the shot hole. Did not the time of producing physical effects in causing distortion, affect the comparative results? He thought that the piezo-electrical gage they had just seen at the Buxton Research Station might be a valuable aid in contributing to the study of shock waves from explosives through air by obtaining time records in connection with impacts.

He also asked if repeated shots in a blast hole in rock did not cause enlargement that might affect results when using a large quantity, say 800 grams, as compared with a lesser quantity of explosive.

Beyling, replying to the first question, said he considered the suggestion of making a time record of the impact of a pressure wave was good, but the experimental difficulties would be great. He had wanted to determine whether the pressure is increased in front of the shot hole and whether there is any connection between the ignition of fire damp and a sudden high pressure as produced by a blown-out shot, and he had tried to determine this by the simplest methods available.

As concerns the enlargement of the borehole, Beyling said that in his flame experiments 10 to 15 shots were fired in successive tests in the same hole and after each test the hole

was carefully measured and examined as to diameter and roughness. Appreciable variation in most instances was not noted. Different results naturally would be obtained in shale or coal.

Wheeler said that Rice's criticism of the use of lead disk to determine pressures because of the time factor should be considered.

Beyling replied that he realized that the lead plate was not an accurate method of measuring the pressure. Although the time factor was not considered, he thought the experiments could be regarded as indicating relative pressures in front of the borehole.

Breyre remarked that from Beyling's experiments, it would appear that the flame was not particularly important in the ignition of fire damp and that pressure also appeared to have little influence. It therefore remained to find out the culprit. He asked whether Beyling could give any idea as to what magnitude of pressure was indicated by a given deformation of the lead disk. He also wished Beyling to amplify his remarks concerning ignition by cartridges freely suspended.

Beyling said he did not preclude the possibilities of ignition of fire damp by the flame or by the compression due to a blown-out shot, but intended to convey the meaning that ignitions could be obtained which were not attributable to either of these causes. Regarding calibrating the lead disks, this had been considered, but it was regarded as an additional complication which for the experiments undertaken had not been thought necessary.

Audibert agreed with the suggestion made by Beyling that the detonation of an explosion takes place in two phases, the first being the travel of the explosion wave and the second the combustion of the gases. He did not believe that combustion is necessarily complete in the detonation wave; in fact, the experiment that he had carried on at Montlucon had definitely proved that it was not so. Beyling had gone further and said that, not only is combustion incomplete, but that certain portions of the explosive itself are not touched and remain behind the explosion wave. He suggested that this could be shown by calculation, that when a certain quantity of explosive burns, a given volume of gas should be formed, whereas the volume actually formed is much less.

With respect to photographs of freely suspended cartridges, they had succeeded at Montlucon in photographing the discharge of cartridges 100 to 200 grams but had shown that the effect on the photographic plate was exactly the same whether the explosive was one which would ignite fire damp or one which would not.

The writer asked what Beyling's conclusions were with regard to the part played by hot gases. He suggested that in Beyling's results, when he showed that certain large flames were combustion was more complete in one instance with rapid cooling due to the expansion than in the other. Furthermore, would not the temperature of the gases immediately beyond the end of visible flame be sufficient to ignite fire damp.

Beyling replied that Rice's suggestions were very interesting. His own experience was that the flame was rarely the cause of ignition. (This had reference to the flame of a permissible explosive).

Wheeler in closing the discussion said he had only one question to ask Beyling and that was, what is "flame." Were the pictures on his photographic plates necessarily of flame.

Beyling said that he no more knew what flame was than Wheeler did, and he could only assume that the luminous image shown on the photographic film was of flame.

Address by W. Payman on the Application of Schlieren Photography in Researches on Explosives

Payman said he had not prepared a formal paper because much of the work he was going to describe had been published or was in the course of publication. After briefly describing

the general methods of testing explosives at the Buxton station, he said as his address would to a large extent supplement Beyling's he would quote two of his remarks: First, they could not hope to have many further improvements in the safety of explosives until they were able to say what it is that causes the ignition if fire damp, and why an explosive sometimes does and sometimes does not ignite fire damp Secondly, Beyling had shown that ignition of fire damp could be obtained when any one of the many and varied possible causes was absent. He (Payman) would add this observation, that any of the suggested causes of ignition - flame, hot gases, compression, solid particles - might cause the ignition of fire damp, but that no particular one is a necessary cause of ignition.

At Buxton they were trying by a photographic method to trace each of the effects obtained when an explosive is fired. They were not content to follow one single effect because they knew that each of them might assist another. Thus a pressure wave may pass through a gas mixture and heat it. It may not heat it sufficiently to ignite the gas, but it makes the task of ignition by flame, hot gases, or solid particles easier when they in turn come in contact with the gas mixture. They were also trying to follow the course of the process.

Payman described and illustrated with lantern slides different parts of the research work on explosives being carried on at the Buxton station and stated that present views of the staff were given in a publication just issued by the Safety in Mines Research Board, which was distributed to members of the conference in advance form.[10]

Payman then briefly described the research by the Schlieren method of the ignition of fire damp by explosives. This method had been developed at the Buxton station, and recently, under the cooperative agreement for research between the board and the United States Bureau of Mines, had been carried on further at the latter's explosives testing station near Bruceton, Pa. A Schlieren apparatus had been installed there by him and W.C.F. Shepherd of the board's staff. The latter had been detailed for that work for a period of 17 months and had just returned to England. Shepherd had been able to take several interesting series of photographs illustrating fire damp-air mixtures ignited by explosives.

The Schlieren method employs a concave metal mirror. A beam of light thrown on it is reflected as a converging beam, the beam is refracted by waves of atmospheric disturbance, such as produced by shock waves, flame, etc., from the explosives under test, and the effects are recorded on a high-speed photographic film.[11]

Discussion of Various Mine Safety Matters

The program of the conference, as originally planned under the auspices of the Safety in Mines Research Board, related wholly to explosives, but at the suggestion of Sir Edward Troup other subjects were discussed informally on the last day of the conference (July 16). Among the problems briefly discussed were (a) the application of stone-dust (rock-dust) for the prevention of coal-dust explosions and (b) the testing of flameproof mining electrical apparatus. It was decided to defer detailed consideration of the problem of coal-dust explosions, including such matters as the use of stone-dust barriers and the most suitable kinds of stone-dusts to use, both for barriers and for general distribution on mine roadways, until another conference, the subjects being too large to deal with during the time remaining. However, the subject of testing the physical condition of stone-dust was briefly discussed.

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10 - Grimshaw, H. C., and Payman, W., The Ignition of Fire Damp by Coal-Mining Explosives; Gallery Experiments: Safety in Mines Research Board, Paper 69, 1931, 45 pp.

11 - Shepherd, W.C.F., The Ignition of Fire Damp by Explosives, A Study of the Process of Ignition: Bull. 354, Bureau of Mines, 1932, 89 pp.

Testing the Tendency of Rock-Dusts to Absorb Moisture and to Cake

The problem involved in the tendency of rock-dusts to absorb moisture and to cake had arisen more or less acutely in Great Britain, Germany, and the United States. In some mines or in parts of mines where at times the mine air is humid or supersaturated, there is a tendency for the rock-dust distributed in the mine passages or in rock-dust barriers to cake and thus not be effective to raise as a dust cloud when an explosion occurs.

In England the question had come up in connection with the Haig Pit (Whitehaven) explosion disaster in the previous winter, and has frequently arisen in coal mines of the United States in the summer time because of the moistening of rock-dust by humid intake air and also in mines where watering is required at the mine faces, as in Utah. This condition also occurs in German mines and has been a matter of testing by Beyling. He subjects rock-dust samples to exposure in mine air for a certain period, and then uses the simple method of blowing a sample held on the palm of one's hand and estimates the relative amount dispersed.

At the Sheffield laboratory (England) a method of dispersal of a stone-dust sample by a suddenly applied jet of air has been tried; also other methods are being tested.

The United States Bureau of Mines has experimented with moistened rock-dust in various ways; tests have been made of the relative absorption of water by samples in tubes standing in water, the relative hardness of the caked dust as determined in a coke-crushing device, and the relative dispersibility of a rock-dust sample after having been spread for some weeks or months in the Bruceton Experimental mine.

The discussion by Wheeler, Beyling, Audibert, and the writer indicates that the problem had several phases: (1) The greater susceptibility of some rock or stone dusts than others to absorb moisture, (2) a difference in the hardness of caking when moistened, (3) a difference in the dispersability of rock-dusts by air waves or by blasts, of an explosion, when these dusts have been previously wet and then dried by the air currents.

It was agreed that none of the empiric methods now employed were satisfactory and it might well be a problem for an international cooperative study to develop test methods that might be standardized and capable of being employed quickly and conveniently by mining companies for the selection of a rock-dust least affected by humidity for use in their mines.

Testing of Mining Electrical Apparatus

As regards mining electrical apparatus, it was agreed by the conferees that it should be possible to draw up a unified code of constructural requirements if not of tests, and that steps should be taken to prepare such a code for consideration.

PLAN OF INTERNATIONAL COOPERATION ON SAFETY IN MINES RESEARCH

The final important decision of the conference was that arrangements for a general international cooperation on safety in mines research should be made on the lines suggested by Sir Edward Troup, and the following proposals were adopted by the delegates, subject to ratification by the organizations concerned:

(1) The arrangements will only relate to health and safety in coal mines.

(2) The general arrangements shall be as between the directors of the respective research stations.

(3) Cooperation will be effected principally by mutual periodic interchange of progress reports on programs of experimental work on health and safety in coal mines undertaken by the research stations.

(4) Such progress reports will be brief, but can be supplemented, if desired, by detailed reports on individual problems.

(5) If convenient, the cooperation will provide for the interchange of personnel, or the allocation of a member of the staff of one research station for work at another; and for the loan of apparatus and the supply of materials.

(6) An important feature of the cooperation to be aimed at, if possible, will be the production of agreed reports on matters of outstanding interest, with all the directors of research as joint authors.

(7) Periodic meetings of the directors of research shall be arranged at each research station in rotation.

On Friday, July 17, the Safety in Mines Research Laboratories at Sheffield were visited, and the conference then dispersed.

INTERNATIONAL INTERCHANGE OF REPORTS AND INFORMATION

Following the Buxton conference informal negotiations were entered into between the directors of the mine safety research work of Great Britain, France, Belgium, and Germany and the director of the United States Bureau of Mines, with reference to interchange of information, this for the present to involve only an exchange of confidential quarterly reports and brief progress reports of different pieces of mine safety and health research.

This proposal was agreed to by the respective directors, and the plan has been operative since January 1, 1932.

Such interchange is of undoubted value; often a specific research may not reach maturity and the findings be not sufficiently assured to permit publication for several years, if at all, or the research may even fail in the results hoped for by the investigator; yet the progress reports on that problem and the data obtained may be of great value to other investigators and may prevent needless experimentation by another station which might otherwise try the same procedure. Unnecessary duplication of work is avoided and money is saved to the respective organizations. Also if the same conclusions on some problem are reached by different ways of approach by two or more of the research stations, the agreement in results unquestionably gives added confidence in making specific recommendations to mining men concerning safety in their mines, and this is manifestly of importance to the whole coal mining industry. Already there have been a number of such instances of agreement in findings obtained through the British-American cooperation in mine safety research.

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SAFETY PROGRESS IN ANTHRACITE AND BITUMINOUS-COAL FIELDS[1]

By D. Harrington[2]

The mining industry of the United States has for many years possessed the dubious distinction of having the poorest accident rate of all of the major lines of industrial endeavor in this country, and until recently there was little or nothing to show that any material improvement might be expected. However, there are now numerous trends indicating that the corner has been turned at last, and it is strongly hoped that the mining industry is likely to follow the railroads, the cement and other industries in operating with occurrence of only a reasonable amount of loss of life and limb through accidents.

The complete accident figures for metal and nonmetallic mineral mining for 1931 are not yet available, but fragmentary data show that 1931 will unquestionably have very low accident rates, probably the lowest in the history of our metal mines. One underground metal mine has produced over 14,000,000 tons of copper ore without a fatality; another metal mining company with an electric shovel operation has handled upwards of 75,000,000 tons of material, copper ore and waste, without the occurrence of a fatality; an iron-ore company which owns six mines (three underground, three open pit), has operated every one for a year or more without a lost-time accident with a considerable number of employees and a large production; and another iron-ore producer with 30 mines and about 3,000 employees has an average of nearly 10 months of operation in 1931 without a lost-time accident. Numerous other wonderful safety records of metal mines in 1931 indicate unmistakably that the record of the noncoal mining industry of the United States as a whole in 1931 will undoubtedly be found to be excellent.

According to the latest relatively complete figures (those of 1930) available for the coal industry, fewer than 650,000 persons are employed in and around the coal mines of the United States, the figures indicating that a little over 493,000 people are engaged in bituminous mining and a little under 151,000 in anthracite mining. Bureau of Mines complete figures show that 2,053 persons were killed and 103,821 non-fatally injured in our coal

1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6671."

2 Chief engineer, safety division, U. S. Bureau of Mines.

mines in 1930; in addition to the 2,063 killed, 122 suffered permanent total disability injuries. The figure of 2,063 killed in 1930 was a substantial reduction from the average of 2,409 killed annually in our coal mines for the past 25 years. Inasmuch as 1930 is the first year in which nationwide figures on nonfatal accidents in coal mining have been collected, it is not possible to give comparative nonfatal accident figures for 1930 as against the average for the past 25 years. The fatal accident rate per million tons of coal mined in 1930 was 3.84, or much lower than the 4.45 average rate for the past 25 years.[3]

Although the fatality rate of 3.84 per million tons produced in 1930 is much better than the 4.45 average rate for the past 25 years, it was not so good as the 3.75 rate per million tons mined for the 5-year period 1926-1930, inclusive, and was but little better than the 3.95 rate for the 5-year period 1921-1925, or the 3.86 rate for the period 1916-1920; hence, but little improvement can be claimed in the fatality record of coal mining up to and including 1930. It remained for 1931 with its numerous and varied vicissitudes (among them the fact that the tonnage of about 438,000,000 was the lowest since 1908) to show the way to real progress in coal-mine safety; the tentative coal-mining fatality rate of 3.27 persons killed per million tons mined in 1931 sets a record not equaled or surpassed in any year in the present century, and probably not equaled in any year in the history of coal mining in the United States. With but about 1,430 persons killed in the coal mines of the United States in 1931 as against 2,063 in 1930 and against an average of 2,409 for the past 25 years, it is seen that in 1931 our coal mines had 633 fewer fatalities than in 1930 and had 979 fewer than the 25-year average annual rate. The previous record of the century for small number of coal-mine fatalities in one year in the United States was that of 1922 when 1,984 were killed; hence, 1931 beats the 1922 record by 554 fewer fatalities, and while 1922 produced nearly 477,000,000 tons of coal against but about 438,000,000 tons in 1931 the fatality rate per million tons mined in 1931 was 3.27, or much lower than the 4.16 rate in 1922. The exact 1931 figures will not be available for some months yet, but it seems perfectly safe now to predict that not only the coal-mine fatality rate per million tons mined, but also the fatality rate per thousand 300-day workers, or per thousand employed or per million man-hours worked, will be lower than in any other year in the history of modern coal mining in the United States.

Alaska, Michigan, and Texas had no fatalities in coal mines in 1931, while every other coal-producing State except Indiana, Montana, North Dakota, and Maryland reported fewer fatal accidents in 1931 than in 1930. Alabama's coal mines had an especially good record with but 25 fatalities in 1931 as against 63 in 1930, 72 in 1929, 67 in 1928, 93 in 1927, 139 in 1926, and 161 in 1925. Other States in addition to Alabama which reduced coal-mine fatalities in 1931 by 50 or more per cent as compared with 1930 are New Mexico, Ohio, Oklahoma, Utah, and Washington. In general, much of the improvement in these States has been due to the fact that they had no explosions in 1931, though they had one or more explosions in 1930.

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3 The rate per million tons mined is by no means a fair standard or even an accurate means of comparing accidents, but unfortunately it is of comparatively widespread use in connection with coal-mining accidents in the United States, and will be used largely in this paper.

One of the reasons for the excellent safety record in coal mining in 1931 was the fact that explosions caused only 86 fatalities as compared with 264 in 1930 and an average of 341 annually from 1922 to 1931, inclusive. From January 28, 1931, to November 3, 1931, or over nine months, the bituminous mines of the United States did not have a major explosion disaster, this unquestionably being the longest consecutive period of immunity from major explosion disasters in the bituminous coal mines of the United States in at least 50 years. While numerous factors contributed to this nine months' avoidance of major explosions in bituminous mines (there was one major explosion in an anthracite mine), undoubtedly during this period rock-dusting prevented at least three major disasters with the possible, even probable, death of 300 or more persons.

The 1931 coal-mining fatality record is the more remarkable because of having been established in a year of acute financial depression. During the past 25 years there have been at least four financial depressions of a major nature: That of 1907-1908, the war depression panic of 1914-1915, the primary post-war panic of 1920-1922, and the present or secondary post-war panic of 1930-1932. Naturally, coal production fell off in each case; in 1908 the tonnage was 409,000,000 as against an average of 451,000,000 for the 5-year period 1906-1910, inclusive; tonnage was 513,000,000 in 1914, compared with 570,000,000 in the previous year and 529,000,000 as the average for the five years 1911-1915, inclusive; tonnage was 477,000,000 in 1922 against an average of 559,000,000 for the 5-year period 1921-1925, inclusive; and in 1931 it was about 438,000,000 against an average of 595,000,000 tons for the 5-year period 1926-1930, inclusive. In all of these panic years except that of 1931, the fatality rate per million tons of production was higher than the average of the 5-year period in which it occurred; 1931, on the other hand, with about 1,430 fatalities (according to present available data) and with approximately 438,000,000 tons produced has a fatality rate per million tons of production of but about 3.27, as against 3.75 for the 5-year period 1926-1930, inclusive. Hence, it will be seen that the excellent rate established in the panic or depression year 1931 does not follow the precedent of having a high accident rate, as has happened in other panic years of the present century.

Bituminous mining in 1931, with a fatality rate of about 2.78 per million tons of coal produced, shattered all known safety records, and for the first time in at least 40 years the rate was below 3.00 killed per million tons produced. Anthracite mining, on the other hand, had a fatality rate in 1931 which sadly handicapped the record of the coal industry as a whole; the fatality rate of 6.38 killed per million tons produced in anthracite mining in 1931 was materially higher than the 6.04 average rate of that industry for the 5-year period 1926-1930, inclusive, or the 5.80 average rate for the 5-year period 1921-1925, inclusive, or the 6.07 average rate for the 5-year period 1916-1920, inclusive. In other words, the excellent accident record of the coal-mining industry in 1931 was due entirely to the fine safety work done in bituminous mining; the anthracite industry was a hindrance rather than a help, with its heavy accident occurrence and rate. The anthracite mining fatality rate per million tons produced was materially higher in 1931 than

the average rate of that industry for the past 15 years; in fact, there were but four years out of the last 15 with a higher anthracite mining fatality rate than that of 1931.

The bituminous mines of Pennsylvania made an excellent record in 1931 with but 209 fatalities with a fatality rate of 2.00, as against 380 fatalities and rate of 6.38 in the anthracite mines. The 1931 fatality rate of about 2.00 per million tons produced by the bituminous mines of Pennsylvania was by far the lowest or best rate for those mines since 1883, when the rate was 2.01; the 1931 rate of 2.00 was much better than the 2.88 average for the 5-year period 1926-1930, or the 2.67 average of the 5-year period 1921-1925, or the 2.66 average of the 5-year period 1916-1920. The 209 fatalities in the bituminous mines of Pennsylvania in 1931 are less than half of the average of 420 fatalities annually in those mines for the previous 20 years. Unquestionably one of the outstanding factors in the reduction of fatal accidents in the bituminous mines of Pennsylvania in 1931 was the avoidance of major explosion disasters. No major explosion (one in which five or more lives were lost) occurred in a bituminous coal mine in Pennsylvania in 1931; there has been no major coal mine explosion disaster in a Pennsylvania bituminous mine since March 21, 1929, or for more than three years -- a record which points the way for the entire coal-mining industry of the United States; in fact, the coal mining industry as a whole may be proud of this exceptionally fine accomplishment, even though it is due entirely to the efforts of those engaged in one manner or other in work in the bituminous mines of Pennsylvania. In addition to avoiding major disasters and having probably an all-time low fatality rate, the bituminous mines of Pennsylvania showed numerous other safety accomplishments in 1931 - not a person was fatally burned by a gas ignition; 18 out of 56 awards of the J. A. Holmes Safety Association for safety performances of the mineral industries throughout the United States for 1931 were given to persons or organizations in bituminous mining in Pennsylvania; 30 new chapters of the Holmes Safety Association were organized and more than 20,000 bituminous miners of Pennsylvania are now members of that organization; in 1931 over 24,000 persons engaged in bituminous mining in Pennsylvania took the full Bureau of Mines course in first-aid or mine rescue work, and 33 mining organizations with 43 mines received 100 per cent certificates for having the entire personnel so trained.

An analysis of carefully kept accident data reveals many facts. For instance, Bureau of Mines statistics show that in the 5-year period 1925-1929, inclusive, the total number of fatalities from explosives in Pennsylvania anthracite mining averaged 48 per year for its approximately 160,000 employees; all of the bituminous mines of the United States, with approximately 525,000 employed, averaged but 46 deaths from explosives; and Pennsylvania's bituminous mines, with about 145,000 employed, averaged but 8 such fatalities yearly. This indicates strongly that there is something defective in blasting practice in anthracite mining, and that concerted action should be taken to remedy a condition that is certainly unsafe. That there are relatively safe methods of coal-mine blasting can be substantiated by the fact that an electric

blasting system with firing of all shots from the surface with all persons out of the mines, has been in use for over 40 years in certain Utah and Colorado coal mines which have produced at least 75,000,000 tons of coal without a fatality from blasting.

In the 5-year period 1925-1929, during which about 160,000 were employed in anthracite mines, an average of 52 persons were killed annually in those mines by falls of face or pillar coal; whereas in the bituminous mines of the United States, in which about 625,000 were employed, only about 88 were killed annually from the same cause; and but 30 were killed by such falls in the bituminous mines of Pennsylvania out of about 145,000 employed; these figures indicate almost a 50 per cent excessive death rate from falls of face or pillar coal in anthracite mines. During 1925-1929, inclusive, falls of roof (coal, roof, etc.) annually killed about 172 persons in anthracite mines and about 175 in Pennsylvania bituminous mines, as against 825 in the bituminous mines of the United States, indicating a more favorable rate from this cause in anthracite than in bituminous mines and even a slightly better rate than the bituminous mines of Pennsylvania in proportion to men employed; however, on the basis of exposure (number of fatalities per million man-hours of exposure) the bituminous mines of Pennsylvania as compared with anthracite mines had a more favorable rate from falls of roof and coal by about 10 per cent. It would appear that much additional precaution should be taken to the curbing of accidents from falling material, but especially from falling face or pillar coal in both anthracite and bituminous mines.

The record for the 1925-1929 period shows that annually an average of six persons met death in anthracite mines from suffocation by mine gases and that but five persons met death from this cause in bituminous mines, which employ about four times as many men as anthracite mines. The number is excessive not only in anthracite but also in bituminous mines, and is indicative of poor ventilation and its accompanying evils -- gas explosions, miner's asthma, and carbon monoxide poisoning from explosives and other fumes; poor ventilation is inexcusable and points to inefficiency as well as tolerance of unsafe conditions.

Anthracite mines had an average of 64 killed annually during 1925-1929 from mine cars and locomotives, as against 70 in bituminous mining in Pennsylvania and an average of 388 for the bituminous mines of the entire country. As to deaths from this cause, the anthracite mines had a materially better record than bituminous mining either in Pennsylvania or in the United States as a whole.

With an average of 42 deaths annually from explosions in anthracite mines during 1925-1929 as against 72 in bituminous Pennsylvania mines and an average of 275 in the bituminous mines of the United States, the anthracite rate from this cause is by far superior to either Pennsylvania bituminous or the bituminous mines of the United States. However, the anthracite mines have the advantage that anthracite dust enters into explosions little, if at all,

so that although individual explosions or ignitions of gas are relatively numerous in anthracite mines, the number of deaths per explosion is much less than in bituminous mines, where dust ignition often takes place in explosions; this fact keeps the aggregate number of deaths from explosions in anthracite to a comparatively low figure, but it does not alter the fact that gas ignitions are far too numerous in anthracite mines, indicating that ventilation is by no means as efficient as it should be.

When the United States Bureau of Mines for the first time assembled nation-wide statistics on nonfatal accidents in coal-mining it was found for 1930 that 644,000 persons engaged in coal-mining suffered 103,821 nonfatal injuries -- approximately one injury to every six persons employed. Anthracite mines, with about 151,000 employees and 32,604 nonfatal accidents, had a much poorer record than bituminous mines, with 71,217 nonfatal accidents to its approximately 493,000 employees; moreover, Pennsylvania bituminous mines with 20,414 nonfatal accidents to its 130,000 employees had a much better showing as to nonfatals than did the anthracite mines of Pennsylvania. For 1930 the nonfatal injury rate per million hours worked was 129.1 for anthracite mines, 103.54 for all of the coal mines of the United States, and 98.15 for the bituminous mines of Pennsylvania; hence, the anthracite mines had a much poorer nonfatal rate on the basis of exposure than the coal mines of the United States as a whole or the bituminous mines of Pennsylvania. Evidently, it should not be very difficult to bring about the reduction of 1,000 accidents in 1932 in the anthracite mines as desired and advocated by the Pennsylvania Department of Mines.

While in 1931, as well as in several years preceding 1931, anthracite mining evidently had not been brought into step with the downward trend of occurrence of accidents, it now appears that the anthracite industry realizes that "it can be done," and in 1932 is proceeding to give its bituminous competitors a merry race. In January, 1932, there were but 15 fatalities in anthracite mines as against 41 in January, 1931; in February, 1932, but 19 as against 42 in February, 1931; and in March, 1932, but 24 as against 25 in 1931; hence, for the first three months of 1932 there were but 58 fatal accidents in anthracite mines as against 108 for the first three months of 1931, a reduction of over 46 per cent, while the tonnage reduction was but about 22 per cent. If the anthracite fatality rate for the first three months of 1932 continues through the year, the anthracite industry will have broken all of its records in safety, not only as to fatality rate but also as to number killed and injured; and instead of having between 400 and 500 killed in 1932, the death list will be less than 250. It also seems likely that the goal of 1,000 fewer accidents in anthracite mines in 1932 as advocated by the Pennsylvania Department of Mines will be achieved.

The age-old excuse that coal mining is inherently unsafe has been proved to be utterly unfounded, and it now appears that the dangers in coal mining are man-made to a far greater extent than they are due to nature. Coal mines have been operated for 30 or more years without a fatality, or for a year or

more without a lost-time accident; coal mines or mining companies which under one management had decidedly bad accident records have been operated with utmost safety when under different management; mine officials have operated supposedly dangerous properties for years with but few if any accidents; and there are numerous instances of coal-mine workers who have been engaged in so-called dangerous types of mining activity for 30, 40, 50, 60 or more years without occurrence of an accident which caused them to lose as much as a day of work.

The very gassy mine is seldom the scene of a bad explosion disaster, and almost invariably it is the so-called non-gassy mine with its small amount of gas which takes the heavy toll of life in explosions in coal mines. In a similar way the mine with the very bad roof conditions which must be carefully handled very often has by far a better record as to fall of roof accidents than has the mine with the so-called good roof but where little attention is given to roof-supporting measures. In other words, our mining people know how to handle in a safe way the worst conditions found in mines, but fail to apply this knowledge when the conditions do not appear to be very bad. If the known precautions for safe handling of very gassy mines, of preventing explosions in very dusty mines, of preventing or at least materially reducing accidents in mines with very bad roof, of safeguarding workers in those mines in which haulage is difficult, as for instance on pitches, or of handling of explosives with relative safety -- if all of these known precautions were taken in <u>all</u> mines, then the accident rate in <u>all</u> mines could easily be brought within the figures which now give so much credit to the cement and railroad industries. This may seem to be an untenable statement, but a few of the hundreds of available facts will be given in support.

Alabama had 42 deaths from falls of roof in 1926 and 61 in 1927, and this situation was considered so bad that a campaign against falls of roof accidents was started by the State inspection force in cooperation with the Alabama Mining Institute (coal operators), the Holmes Safety Association (operators and mine workers), and the United States Bureau of Mines. As a result there were but 28 such accidents in 1928, 28 in 1929, 34 in 1930, and 8 in 1931. Similar action was taken against gas ignitions after the death of 42 persons in Alabama's coal mines from this cause in 1926, and the deaths from gas ignitions since that year have been 3 in 1927, none in 1928, 12 in 1929, 8 in 1930, and 3 in 1931. Contacts with electricity for several years were giving a much higher death rate in Alabama's coal mines (many of which are in such low coal as to make the trolley-wire hazard more difficult to avoid) than in almost any other State in the Union; after some agitation a campaign was started against this type of accident, and in 1931 Alabama coal mines had but 2 fatalities from electricity as against 11 in 1930, 11 in 1929, 13 in 1928, 9 in 1927, and 18 in 1926. In every one of these cases in Alabama, the bad record was pointed out by the State inspection force; this aroused the mining people, and all concerned openly entered into a concerted movement with the avowed intention of stopping a particular type of fatal accident and <u>in every case succeeded</u>. As a result of these tactics the

coal mines of Alabama produced 470,000 tons of coal per fatality in 1931 as against but 155,000 in 1926; and while Alabama's coal tonnage had decreased 43 per cent in 1931 as compared with 1926, its fatal accidents in coal mines had decreased 81 per cent.

Coal mining, especially in its attitude toward safety, has been in a rut for many years, but there is no good reason why it can not extricate itself. However, this can be done only through intensive, continuous, intelligently applied hard work. Logically the first step is to know the facts as to occurrence of accidents, including not only fatals, but also all types of nonfatal accidents (serious, trivial, lost time, no lost time, compensable, etc.); and above all it is advisable to get the facts as to the cost of these accidents, not only in life and limb as well as in sickness and in misery, but also as to the cold-blooded dollars and cents outlay demanded through occurrence of accidents. An analysis of each accident is of utmost value, whether or not it results in a personal injury, but an analysis of all of the accidents for a year and a totalling of the dollars and cents of the accident cost as against tons of coal produced will usually speak much louder than volumes of word descriptions of the horrors or miseries entailed through accidents. There is good reason for the belief that cost of accidents in coal production is at least 10 per cent of the mine cost, and there is equally good reason to believe that at least half of this cost can be eliminated with reasonable care; and if half of the accident cost were eliminated from many mines, most of these mines would have a black rather than a red balance. The management of one moderately large producing coal mine a few years ago found that accident compensation alone cost more than $200 for every day the mine operated during the year; this figure was so appalling that stringent measures were put into effect for the prevention of accidents, and within two years the compensation cost per day the mine worked was reduced to less than $10, and the daily tonnage remained at practically the same figures as during the period of high compensation cost.

So many mining organizations, coal as well as metal, are now proving that mining can be conducted safely and efficiently at the same time, and that safe operation automatically brings increased efficiency with lowered costs, that it is absurd to try to convince well-informed people that mines are inherently unsafe or that accidents must occur wherever mining operations are under way. However, safety is not to be secured by the waving of a magic wand or by merely wishing for it; certain well-defined underlying factors must be injected into operation of mines and kept constantly in effect before safe operation can be achieved. A few of these factors are: The management including the "higher ups" must get sincerely and actively and personally behind the safety movement; means must be put into effect to bring about the active interest and cooperation of the mine officials as well as the workers in the avoidance of accidents (and very frequently this is decidedly difficult of achievement and bonuses or some similar inducement must be given either to workers or to officials or to both); every mine should have an actively functioning safety organization with monthly safety meetings at least and preferably with

participation by workers; all persons in every mining organization should be well trained in first aid; every mine should have a set of safety rules, preferably in a printed pamphlet, so it may be at all times available to every employee, and these rules should outline in simple language the minimum safety requirements of the management and of the State law, these rules to be revised from time to time to keep pace with changing conditions and to correct weak points; above all things every mine official should use the utmost care to follow out the safety rules at all times, as it is difficult to find anything more demoralizing in trying to bring about real safety than to have workers see safety rules violated or disregarded by officials; as previously indicated every accident whether or not an injury results, should be studied to ascertain the cause and the probable method of prevention of recurrence; and at intervals data on accidents should be compiled and carefully studied in an effort to eliminate the causes of those of most frequent occurrence or of greatest seriousness in injury or in cost. One of the most fruitful aids in bringing about increased safety in mining is to keep in touch with safety performances of other mines, particularly mines in other regions, with the idea that whatever one operator can accomplish can be done by any other wide-awake operator. And in this it may be well to call attention to the fact that many mines, including a number of coal mines, are now trying to avoid not only fatalities, and permanent disability accidents and serious accidents, but also any kind of lost-time accidents; and numbers of both coal and metal mines have operated a year or more with production of hundreds of thousands of tons of coal or ore without occurrence of any accident which would prevent the victim from returning to his usual work on the following day.

The mine inspectors of the various States can do a valiant service to safety in mining by prompt issuance in mimeographed or similar form of fairly full details concerning not only fatal but also certain nonfatal accidents which are abnormal or the occurrence of which conveys a lesson of value to others. Some of the State inspection forces are now doing excellent work in this respect, and it is to be hoped that others will adopt the practice, possibly with variations or extensions.

To achieve maximum results in the safe operation of mines, there must be the closest cooperation by all of those engaged in the mining industry, including the mine operator and his officials, the mine workers, the State inspection force, and the forces of the United States Bureau of Mines, or others known to be striving to bring about safe operation of mining properties. Without such wholehearted cooperation, real safety in mining will be difficult to attain. It would also appear that one of the main functions of the State inspection forces in addition to that of trying to enforce the existing safety laws should be the dissemination of up-to-date data on safety in mining, even when that information is in addition to or beyond the bare requirements of State law or State regulations.

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While there is ample room for rejoicing over the downward trends of accident rates, coal mining can by no means consider that its present accident-prevention activities meet all its needs, and this refers to bituminous as well as to anthracite mining. No thinking coal-mining man can afford to be complacent when confronted with figures showing that both as to accident frequency and accident severity the coal-mining industry has by far the worst record of all the major industrial activities in the United States. And the bituminous coal mines can not throw the burden of this bad record wholly upon the shoulders of the anthracite industry, for the bituminous rates also are much worse than those of other major industrial pursuits. Moreover, our national pride should not be forced longer to be confronted with statistics showing that even with the improved rates established in 1931 our coal-mine workers have an occupation two, three, or four times as hazardous as have the coal-mine workers of other countries which produce coal in quantity.

1932 SAFETY RECORD

1931 was a record year in coal-mine safety, but if the first three months of 1932 are any criterion, it is likely that 1932 will have even a better safety record. The Bureau of Mines figures for coal-mine fatalities for the months of January, February, and March, 1932, show but 326 fatalities--a much more favorable record than the 411 for those months in 1931. Both anthracite and bituminous mining operations are participating in the good work; anthracite mines have had but 60 fatalities in 1932 up to May 1, as against 108 for the same period last year, and bituminous mines have had 266 as against 303.

States which have materially bettered their safety record for the first three months of 1932 as against the same months of 1931 are Indiana, Ohio, Pennsylvania (both anthracite and bituminous), West Virginia, and Wyoming. States which have maintained or slightly improved a good record set in 1931 are Alabama, Alaska, Colorado, Illinois, Kentucky, Missouri, Montana, Tennessee, Texas, Utah, and Washington.

Virginia has a decidedly poor record for the first three months of 1932 with 54 fatalities against but 5 for the similar period of 1931. Other States which have a poorer record so far in 1932 than for the similar period of 1931 are Arkansas, Iowa, Kansas, Maryland, Michigan, New Mexico, North Dakota, and Oklahoma. Alaska, Montana, and Texas have had no fatalities so far in 1932.

If major disasters can be avoided, and they can be if anything like reasonable care is taken, 1932 should have a better coal-mine safety record than any year in the modern history of coal mining in the United States.

I. C. 6672 December, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

TEN YEARS OF FATAL ACCIDENTS AND TWO YEARS OF ACCIDENT COSTS IN INDIANA COAL MINING

BY

C. A. Herbert

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I.C.6672,
December, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

TEN YEARS OF FATAL ACCIDENTS AND TWO YEARS OF ACCIDENT COSTS IN INDIANA COAL MINING[1]

By C. A. Herbert[2]

The Workman's Compensation Law of Indiana, enacted in 1915, amended in 1917, 1919, 1923, and 1927, and in 1929 named the Indiana Workmen's Compensation Act of 1929, provides for a weekly compensation for disability due to accidents of 55 per cent of the average weekly wage of the employee, to start on the eighth day of disability. Compensation is computed from a maximum and minimum weekly wage of $30 and $16, respectively. Thus, the maximum weekly compensation is $16.50 and the minimum $8.80; the total compensation paid is not to exceed $5,000.

In addition to the compensation the employer is required to pay all medical, surgical, hospital, and nurse fees for a period of 30 days and may be required to pay such fees for an additional 30 days at the direction of the Industrial Board of Indiana. In the case of fatalities the employer must contribute $100 toward funeral expenses.

The law also stipulates that "the employer shall keep a record of all injuries, fatal or otherwise, received by his employees in the course of their employment," also that the employer shall report within one week of their occurrence all accidents that cause either the death of the employee or the loss of more than one day's work. However, no standard form is furnished or required by the industrial board on which the employer shall keep a record of accidents, neither is he required to report the time of return to work of the injured employee unless the period of disability exceeds seven days and the accident thus becomes compensable. Hence, no record is kept of noncompensable accidents other than the first report sent to the industrial board by the employer.

IMPORTANCE OF KEEPING RECORDS

If the employer were required to keep a complete and concise record of all accidents, in duplicate, on a special form either furnished or required by the board, giving all of the essential data in regard to the accidents, and if he were required to submit a copy of this record to the board at the end of the fiscal year, the data thus assembled would not only be of material assistance to the board, particularly in making up its annual report, but would also furnish information of much value in the study of accident causes and what may be done to remove them.

Those industries that have made marked progress in reducing accidents keep such records, as it is only through a full knowledge of how accidents occur that intelligent steps can be taken to prevent their recurrence. Unfortunately, the mining industry as a whole in Indiana has not kept pace with many other industries in reducing accidents, and it is urged as an aid to improvement that a thorough and complete record of mining accidents be kept.

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6672."

2 - Supervising engineer, U. S. Bureau of Mines Safety Station, Vincennes, Ind.

Accidents to men who have not returned to work because of the injury, or accidents for which claims have not been settled at the close of the fiscal year, should be carried forward on the next year's report.

Following is a suggested form of report (fig. 1) that might be used in connection with accidents in mining, and doubtless a similar general form could be made up applicable to other industries. These reports should be filed separately as to industry and alphabetically as to company. The sheet could be 14 by 20 inches and the columns headed "Cause of injury" and "Nature of injury" each about 4 inches wide.

Table 1 lists the fatal accidents by causes for the 10 fiscal years ending September 30, 1922 to 1931.

Table 1.- Fatal accidents by causes in Indiana coal mines for the fiscal years ending September 30, 1922 - 1931, inclusive

Cause	1922	1923	1924	1925	1926	1927	1928	1929	1930	1931	Total	Per cent of total
Underground:												
Falls of roof	13	25	30	22	22	16	19	19	16	13	195	35.1
Falls of coal	2	2	2	1	4	1	-	1	2	3	18	3.2
Mine cars	7	12	2	3	11	4	3	3	4	3	52	9.4
Locomotives	-	10	7	4	1	2	2	1	-	3	30	5.4
Electricity	1	1	1	2	1	1	2	4	-	5	18	3.2
Gas and dust explosions	6	9	5	65	6	43	-	2	1	29	166	29.9
Mine fires	-	-	4	-	-	-	1	-	-	-	5	.9
Explosives	2	5	-	-	1	-	1	2	-	1	12	2.2
Mining machines	-	-	-	-	-	1	-	-	-	-	1	.2
Loading machines	-	-	-	-	-	-	-	1	-	-	1	.2
Suffocation (other than in fires or explosions)	1	1	-	-	-	-	-	1	-	1	4	.7
Animals	3	-	-	-	-	1	-	-	-	-	4	.7
Falling objects	-	-	-	1	-	-	1	1	-	1	4	.7
Miscellaneous	3	-	-	-	-	-	1	2	-	1	7	1.2
Shaft:												
Falling down shaft	1	1	1	-	-	-	2	1	-	2	8	1.5
Cage	-	5	-	2	1	2	-	4	-	1	15	2.7
Material falling down shaft	-	-	1	-	-	-	-	-	-	-	1	.2
Miscellaneous	-	-	1	-	-	-	-	-	-	1	2	.4
Surface:												
Railroad cars	-	1	-	1	1	1	-	-	-	1	5	.9
Machinery	-	-	1	-	-	2	-	-	-	-	3	.5
Falling material	-	1	-	-	-	-	-	-	-	-	1	.2
Falling persons	-	2	-	-	-	-	-	-	-	-	2	.4
Miscellaneous	-	1	-	-	-	-	-	-	-	-	1	.2
Total	39	76	55	101	48	74	32	42	23	65	555	100.0
Coal produced, million tons	14.95	25.36	19.21	17.69	18.50	16.17	11.25	12.07	10.89	9.06	-	-
Fatalities per million tons mined	2.61	3.00	2.86	5.71	2.60	4.6	2.84	3.48	2.11	7.16	3.70	-
Average for United States 1911 to 1928	3.77											
Fatalities per 1000 employed	1.35	2.43	2.01	5.09	2.68	3.39	2.11	3.48	2.14	6.53	3.12	-
Average for United States 1911 to 1928	3.08											

COAL—MINE ACCIDENT REPORT FOR FISCAL YEAR OCTOBER 1, 19___ TO SEPTEMBER 30, 19___

	Name	Address
Coal company or proprietor	- - - - - - - - - - - - - - -	- - - - - - - - - - - - - - - - - - -
Mine	- -	- - - - - - - - - - - - - - - - - - -
Insurance carrier	- - - - - - - - - - - - - - - - - - -	- - - - - - - - - - - - - - - - - - -

Date of Injury

Name of person injured	Address of person injured	Occupation	Age	Number of dependents	Place in mine or on surface where injury occurred	Cause of injury	Nature of injury if fatal, give date of death	Degree of injury*	Date of return to work	Days lost	Compensation paid	Medical and hospital or funeral expenses	Total expenses	Name of foreman in charge of injured
									Total					

* Indicate by symbols as follows:
T. Temporary disability
P.P. Permanent partial disability
T.D. Total disability
F. Fatal

Figure 1.— Suggested form of mine-accident report

The foregoing table includes 31 fatalities in small mines employing less than 10 men, over which the inspection department has no jurisdiction and which are not shown in the annual reports of the Indiana Department of Mines and Mining; for this reason the totals shown for the various years do not always correspond to the totals given in those annual reports. The foregoing data were obtained from the files of the department through the courtesy of Albert C. Dally, chief mine inspector.

Of the 166 men killed by gas and dust explosions, 116 were killed in three major disasters: Namely, 51 in the City mine, February 20, 1925; 37 in the Francisco mine, December 9, 1926; and 28 in the Little Betty mine, January 28, 1931. These explosions were attributed to ignition of gas by open lights. All three could doubtless have been prevented had there been better supervision and better ventilation, or had permissible lights and permissible equipment been in use; without doubt they could have been limited in extent, with a great reduction in the loss of life, had the mines been thoroughly rock-dusted.

Of the remaining 50 men killed by explosions, 40 were killed in so-called shot-firer's explosions in mines where the coal is shot from the solid with black blasting powder, and all the explosions were due to overcharged and improperly placed shots. There is little or no excuse for fatalities from such a cause, as it could be eliminated if the coal were undercut and shot with permissible explosives, as is now being done in the majority of mines in Indiana.

Ten men were killed in minor gas explosions due to the ignition of gas by open lights. As in the case of the three major disasters, these deaths could doubtless have been prevented had there been better supervision and better ventilation, or had the mines been using closed lights and permissible machinery.

Of the fatalities attributable to mine fires, four were due to the ignition of gasoline by an open light in a small mine employing fewer than 10 men. The ignition occurred while the tank on a small oil engine was being filled.

All of the fatalities due to explosives were caused by the improper handling of black blasting powder, again pointing to the desirability of undercutting the coal and using the much safer permissible explosives.

During the 10-year period 1922-1931 there were 18 fatalities due to electricity; 11 were caused by contact with trolley wire and 7 by contact with other circuits. By taking greater care in guarding the trolley wire and the other circuits, accidents from this source could be largely eliminated. In many cases the use of storage-battery equipment would tend to eliminate accidents and would also add to efficiency.

Eight fatalities resulted from falling down shafts and were doubtless due mainly to lack of proper illumination at surface landings and to lack of safety gates at these landings. The United States Bureau of Mines in studying mine illumination has observed the beneficial results of good lighting, and strongly recommends adequate illumination as an aid to accident prevention. The use of up-to-date electric cap lamps would be a long step forward in increasing the efficiency of the miners' lighting in Indiana's coal mines.

Of the fatalities charged against cages, the majority were due either to being caught under the cage while working in the sump or to falling off the cage while it was being hoisted or lowered. These happenings show the necessity of taking every precaution against accidentally lowering the cage on men working beneath it, and also the desirability of having the cage equipped with a gate to be kept closed while men are being lowered or hoisted.

Another point of interest brought out by a study of the fatalities during the past 10 fiscal years is that 31, or 5.6 per cent, occurred in small mines employing less than 10 men. The State mining law does not apply to mines employing less than 10 men; hence, the Indiana Department of Mines has no jurisdiction over such mines and no inspection is made of them.

While there is no way of determining the number of men employed in these small mines, it can not be large, and as they operate only during the winter months to supply coal to the domestic market, it is quite probable that the fatality rate in them is far greater than in the larger mines. In fairness to the men working in small mines it would appear that the mining law should be so amended as to be applicable to all mines, regardless of the number of men employed, so that the men working in the small mines might have the protection the mining law would afford, as well as the benefit of regular inspection by State inspectors.

The one fatality listed under suffocation in 1931 was that which occurred when a boy 11 years old climbed down the stairway of the escape shaft of an abandoned mine and was overcome by black damp; this is only one of numerous similar occurrences that have taken place in various parts of the country in recent years and clearly points to the necessity of guarding the surface openings to all old and abandoned mine workings.

Table 2 lists the fatalities for the 10-year period by occupation. In the column headed "Employees, approximate per cent" an attempt is made to compute as near as possible the percentage of each class of employment with respect to the total number employed in and around the mines. The data from which these figures were computed represent the actual employment at approximately 50 per cent of the mines of the State and are believed to be substantially correct. There is, however, a wide variance in the employment at mechanical and hand-loading machines, and doubtless a complete report from all the mines would change the figures somewhat; for instance, the percentage for drillers is believed to be somewhat high.

Using these figures on the percentage of employment for the various occupations, together with the number of fatalities for each occupation as a basis, the relative hazard for each occupation on the basis of 100 has been computed and is shown in the column headed "Relative hazard of occupation." As shot firers have the highest fatality rate for the number employed in this capacity, they were assigned the number 100.

The average age of those killed during this 10-year period was found to be 39. As most miners begin work in the mines before they are 18 years of age, it is reasonable to assume that these men had had an average of nearly 20 years of mining experience. The fatalities can not therefore, be charged to lack of experience, but rather to the lack of making proper use of the knowledge that should have been gained in their working years. It will be noted that the shot firers, constituting only 0.8 per cent of those employed, had 7 per cent of the fatalities in Indiana coal mines over the 10-year period 1922-1931, inclusive; this record indicates that shot firing practices are anything but safe. The various underground bosses constituted 2.1 per cent of the personnel, but supplied 4.3 per cent of the fatal accidents for the 10-year period, indicating either that "bossing" is a dangerous job or that the bosses are not as careful as they should be. Trip riders and drivers, constituting 5.4 per cent of those employed, had 11.8 per cent of the fatalities, indicating that much greater care should be taken in these occupations.

ACCIDENTS

The annual reports of the Industrial Board of Indiana give little information on mine accidents. However, through the courtesy of Judge Rosco Kiper, chairman of the board, the United States Bureau of Mines was given access to its files and as complete data as possible were obtained on compensable accidents (an accident causing more than 7 days disability is compensable), for the fiscal years ending September 30, 1930 and 1931. No figures were collected on noncompensable accidents, as the employer is not required to report the date of return to work in such cases and therefore no data could be obtained on the time lost for this type of accident. In addition, as the files are not segregated as to industry, it would have required at least 30 days time to collect the available data on these accidents, and it was not believed that the information would be of sufficient value to warrant the additional expenditure in collecting it.

Table 2.- Fatal accidents by occupation in Indiana coal mines for the fiscal years ending September 30, 1922-1931

Occupation	1922	1923	1924	1925	1926	1927	1928	1929	1930	1931	Total	Fatalities per cent of total	Employees approximate per cent	Relative hazard of occupation
Miner	20	33	27	50	17	41	18	14	14	18	252	45.4	50.1	10
Mining machinemen	1	2	4	8	5	4	2	3	2	11	42	7.5	5.1	17
Loading machinemen	-	-	1	-	-	-	-	-	1	11	13	2.3	4.3	6
Motormen	-	1	-	7	1	2	2	3	-	2	18	3.2	3.8	9
Trip rider	1	8	6	4	3	1	2	5	2	3	35	6.3	3.4	21
Driver	6	7	3	3	5	4	-	2	-	1	31	5.5	2.0	31
Tracklayer	2	-	-	5	1	1	-	1	-	6	16	2.9	3.1	11
Laborer	-	5	4	5	3	3	2	-	-	-	22	3.9	2.5	16
Trapper	1	-	1	-	1	2	-	-	-	-	5	.9	.5	21
Superintendent	-	-	-	-	1	-	-	2	-	-	3	.5	.5	11
Mine boss	-	-	-	1	1	-	2	2	1	1	8	1.5	.5	34
Room boss	-	-	2	-	3	-	-	-	-	3	8	1.5	.8	21
Motor boss or parting boss	1	-	1	-	-	-	-	-	-	-	2	.4	.2	23
Fire boss	-	1	-	-	-	1	2	-	-	1	5	.9	.6	17
Surveyor	-	1	-	-	-	-	-	-	-	-	1	.2	.2	11
Shot firer	7	8	2	13	2	4	-	1	1	1	39	7.0	.8	100
Timbermen	-	1	-	1	-	2	-	3	1	1	9	1.7	1.5	13
Shot runner	-	1	-	-	-	-	-	-	-	-	1	.2	.2	11
Cager	-	2	1	-	-	4	1	1	-	-	9	1.7	.8	24
Electrician	-	1	1	2	2	-	-	1	-	2	9	1.7	1.5	13
Pumper	-	-	-	-	1	1	-	1	-	-	3	.5	.5	11
Driller	-	-	-	1	-	1	-	3	1	1	7	1.2	1.7	8
Miscellaneous	-	-	-	-	-	-	-	-	-	1	1	.2	6.1	0.3
Top boss	-	1	1	-	-	-	-	-	-	1	3	.5	.4	14
Hoisting engineer	-	-	-	-	-	2	-	-	-	-	2	.4	.8	6
Top labor	-	4	1	1	2	1	1	-	-	1	11	2.0	8.1	3
Total	39	76	55	101	48	74	32	42	23	65	555	100.0	100.0	-

Tables 3, 4, 5, and 6 have been prepared from the data collected on compensable accidents. Table 3 gives the number of accidents, the days lost, and the compensation paid by occupation for the year ending September 30, 1930; also the percentage of total accidents, percentage of days lost, and percentage of compensation for each occupation. The column "Employment, per cent" shows the approximate percentage of employees in each occupation with respect to the total employed and, as previously explained, was prepared from employment figures obtained from approximately 50 per cent of the mines of the State.

The column "Relative accident hazard" shows the relative hazard to accidents for each occupation for the year, taking into consideration the percentage of total employment for each occupation and the percentage of total accidents received by each occupation. Drivers, having the highest frequency, are shown as 100, while the other occupations are shown in relation to 100.

The column "Relative severity hazard" shows the relative severity of accidents for the various occupations on the basis of 100 and was computed from the percentage of employment and percentage of total days lost for each occupation. In arriving at the days lost in this as well as in the other tables, the standard evaluation of 6,000 days for each fatality was used.

Table 4 is similar to Table 3 and gives the data for the fiscal year ending September 30, 1931. The great dissimilarity in the "Relative accidents hazard" and the "Relative severity hazard" for the years 1930 and 1931, may be explained, in part at least, as being due to the explosion which occurred in a mechanically operated mine during the year 1931.

Tables 5 and 6 give the number of accidents, the days lost, and the compensation paid, by cause, for the years 1930 and 1931, respectively; also the percentage of total accidents, the percentage of total days lost, and the percentage of total compensation paid for each type of accident are shown.

The dissimilarity in the percentages shown in Tables 5 and 6, as in Tables 3 and 4, are largely due to the explosion in 1931 in which 28 lives were lost.

Items 9 to 19, inclusive, of Tables 5 and 6 are chargeable to haulage and taken in the aggregate give a total of 186, or 29.5 per cent of all compensable accidents, 26.7 per cent of the total days lost, and 31.7 per cent of the compensation paid, for the year 1930; and 304, or 27.4 per cent of the total accidents, 12 per cent of the days lost, and 15.9 per cent of the compensation paid, in 1931. Items 9, 17, and 18 are chargeable to lack of clearance, and in 1930 account for 37 or 5.9 per cent of the total compensable accidents, 3.4 per cent of the time lost, and 5.9 per cent of the compensation paid; for 1931 these same three items were responsible for 62 or 5.6 per cent of the compensable accidents, 1.01 per cent of the total time lost, and 1.92 per cent of the compensation paid.

Items 14 and 16, caught between cars and caught between car and locomotive, are mainly chargeable to coupling or uncoupling cars while in motion. In 1930 these two items accounted for 52 or 8.3 per cent of total compensable accidents, 13.3 per cent of the days lost, and 13.3 per cent of the compensation paid; in 1931 they were responsible for 68 or 6.1 per cent of the total accidents, 2.8 per cent of the days lost, and 4.0 per cent of the compensation paid.

The annual reports of the industrial board for the fiscal years 1930 and 1931 show that a total of 2,303 lost-time accidents was reported in 1930 and of 2,248 in 1931, a decrease of 2.3 per cent. From the figures given in the annual reports of the Indiana Department of Mines and Mining it is found that the tonnage for 1931 was 16.8 per cent less and the days of exposure 16.7 per cent less than in 1930. On considering these latter facts it will be seen that accidents, or at least the accident rate, decidedly increased in 1931 over 1930.

The totals on Tables 5 and 6 show that there were 629 compensable accidents in 1930 and 1,113 in 1931, an increase of 76.9 per cent; also, there were 127,659 lost days as a result of compensable accidents in 1930, and 342,186 lost days in 1931, an increase of 168 per cent; at the same time there was a decrease of 16.8 per cent in tonnage and 16.7 per cent in days of exposure in 1931 as compared with 1930.

The total amounts paid in compensation represent only part of the total direct cost of accidents, as there must be added the cost of hospitalization of all accidents, both compensable and noncompensable. Actually, the direct cost of accidents to the operator is equal to the amount of indemnity insurance premium he is obliged to pay.

The average compensation insurance rate in Indiana is approximately $5.50 per $100 of pay roll, and as the annual reports of the Department of Mines and Mining give the total wages paid in 1930 as $13,400,204 and in 1931 as $10,713,103, it will be seen that the cost of compensation insurance, or in other words, the cost of accidents to the operator, was approximately $737,000 in 1930 and $589,200 in 1931.

Table 3.- Compensable accidents by occupation for the year October 1, 1929, to September 30, 1930

Occupation	Accidents		Days lost		Compensation paid		Employment, per cent	Relative accident hazard	Relative severity hazard
	Number	Per cent	Number	Per cent	Amount	Per cent			
Miners	281	44.6	58,428	45.8	$80,468.92	42.0	50.1	26	14
Machinemen and helpers	68	10.8	10,578	8.3	15,945.35	8.3	5.1	39	25
Motormen	25	4.0	6,377	5.0	16,935.93	8.9	3.8	31	21
Loading machinemen and helpers	28	4.4	8,482	6.7	12,877.82	6.8	4.3	30	25
Trip riders	35	5.5	13,681	10.7	16,099.82	8.4	3.4	49	50
Drivers	43	6.8	2,386	1.9	5,699.66	3.0	2.0	100	15
Tracklayers	18	2.9	1,298	1.0	3,805.35	2.0	3.1	42	5
Timbermen	24	3.8	7,474	5.9	9,205.81	4.8	1.5	74	63
Jerrymen or laborers	21	3.3	948	.7	2,077.90	1.1	2.5	39	4
Trappers	1	.2	14	.01	8.80	.0	.5	12	0.3
Cagers	6	.9	125	.1	185.21	.1	.8	33	2
Electricians	4	.6	124	.1	261.87	.2	1.5	12	1
Pumpers	3	.5	63	.05	99.00	.05	.5	29	1.6
Drillers	11	1.7	7,581	6.0	9,469.09	5.0	1.7	29	56
Bratticemen	2	.3	201	.1	664.71	.35	.5	18	3
Shot runners	–	–	–	–	–	–	.2	–	–
Shot firer	2	.3	6,399	5.0	5,991.50	3.0	.8	11	100
Spraggers and couplers	12	1.9	1,365	1.1	3,353.15	1.8	.6	33	27
Superintendents	1	.2	63	.05	132.00	.1	.5	12	1.6
Mine boss	1	.2	39	.02	75.43	.04	.5	12	0.6
Motor or driver boss	–	–	–	–	–	–	.2	–	1
Fire boss	1	.2	77	.05	165.00	.1	.6	10	1
Room boss	1	.2	180	.1	429.00	.2	.8	7	2
Surveyor	–	–	–	–	–	–	.2	–	–
Other underground	3	.5	161	.1	2,589.75	1.4	5.0	3	0.3
Top boss	1	.2	13	.01	14.16	.01	.4	16	0.4
Weighman	–	–	–	–	–	–	.5	–	–
Blacksmith and helper	1	.2	16	.01	21.21	.01	.9	6	.1
Carpenters	1	.2	30	.02	1,335.00	.6	.4	16	.8
Hoisting engineers	1	.2	35	.02	66.00	.03	.8	7	.4
Firemen	1	.2	30	.02	16.50	.01	.6	10	.6
Flat trimmers	10	1.6	350	.24	667.19	.3	2.0	23	1.6
Other top labor	23	3.6	1,141	.9	2,596.53	1.4	3.7	29	4
Totals	629	100.0	127,659	100.0	191,257.66	100.0	100.0	–	–

Table 4.- Compensable accidents by occupation for the year October 1, 1930, to September 30, 1931

Occupation	Accidents		Days lost		Compensation paid		Employ-ment,	Relative accident	Relative severity
	Number	Per cent	Number	Per cent	Amount	Per cent	per cent	hazard	hazard
Miners	490	44.0	44,638	13.0	$73,069.14	19.6	50.1	29	3
Machinemen and helpers	86	7.7	47,685	14.0	48,137.60	12.9	5.1	50	33
Motormen	39	3.5	14,664	4.2	18,627.02	5.0	3.8	30	15
Loading machinemen and helpers	106	9.5	105,514	30.8	92,936.43	24.9	4.3	73	100
Trip riders	67	6.0	21,327	6.2	22,835.78	6.1	3.4	58	22
Drivers	67	6.0	9,947	2.9	10,759.31	2.8	2.0	100	17
Tracklayers	39	3.5	27,674	8.1	29,510.44	7.9	3.1	37	32
Timbermen	24	2.1	1,057	.3	2,896.91	.7	1.5	47	2
Jerrymen and laborers	35	3.2	1,292	.38	4,011.55	1.7	2.5	47	2
Trappers	2	.2	147	.04	229.15	.06	.5	13	1
Cagers	12	1.0	581	.17	1,178.79	.3	.8	40	3
Electricians	13	1.0	20,306	6.0	20,453.58	5.4	1.5	22	49
Pumpers	9	.8	6,142	1.8	5,252.71	1.4	.5	53	44
Drillers	22	2.0	6,962	2.3	7,411.59	1.9	1.7	39	17
Bratticemen	4	.4	206	.06	419.57	.1	.5	27	1
Shot runners	1	.1	164	.04	742.50	.2	.2	17	2
Shot firers	4	.4	6,111	1.8	5,213.14	1.4	.8	17	27
Spraggers and couplers	7	.6	211	.06	568.08	.15	.6	33	1
Superintendents	2	.2	66	.02	122.57	.04	.5	13	0.5
Mine boss	4	.4	6,181	1.8	5,429.50	1.5	.5	27	44
Motor or driver boss	2	.2	258	.07	575.14	.15	.2	33	4
Fire boss	4	.4	6,135	1.8	5,318.80	1.4	.6	23	37
Room boss	5	.5	12,089	3.5	10,280.56	2.8	.8	17	54
Surveyor	1	.1	287	.08	660.00	.17	.2	17	5
Other underground	3	.3	139	.03	275.71	.07	5.0	2	0.1
Top boss	-	-	-	-	-	-	.4	-	-
Weighman	1	.1	168	.04	396.00	.11	.5	7	1
Blacksmith and helper	6	.5	188	.05	360.00	.09	.9	20	0.7
Carpenters	4	.4	117	.03	233.35	.06	.4	33	0.9
Hoisting engineers	1	.1	30	.01	288.75	.07	.8	3	0.1
Firemen	5	.5	89	.02	127.29	.03	.6	23	0.4
Flat trimmers	18	1.6	451	.1	752.40	.2	2.0	27	0.6
Other top labor	30	2.7	1,360	.3	3,175.81	.8	3.7	24	.1
Total	1,113	100.0	342,186	100.0	372,249.17	100.0	100.0	-	-

The annual reports show a total coal-mine production for the State of 10,887,499 tons in 1930 and 9,056,810 tons in 1931. Dividing these tonnages into the approximate premium costs for these same years gives an accident cost of 6.7 cents per ton in 1930 and 6.4 cents in 1931; this cost includes only the direct payment in compensation and the medical and hospital cost, but does not include any of the multitude of indirect costs such as loss of tonnage, loss of time of workers other than those injured, and other items of similar nature.

There is no definite information available on the number of all accidents, lost time and no lost time, compensable and noncompensable, but from information that could be obtained it is believed that lost-time accidents may be roughly estimated as one-third of all accidents, and compensable accidents as one-sixth of all accidents.

If we ignore the cost of no lost-time accidents it is found by dividing the approximate premium costs for 1930 and 1931 by the number of lost-time accidents given in the annual reports of the industrial board for these same years, that the cost per lost-time accident to the operator was $320 in 1930 and $262 in 1931.

These figures, as well as those on the cost of accidents per ton of coal produced, show that the cost of accidents to the operator was less in 1931 than in 1930. The actual cost of accidents, however, was much higher in 1931 than in 1930, the difference being absorbed by the insurance carriers. Unless the accident rate is decreased, the premium rate will of necessity have to be increased, as the insurance carriers can not continue the insurance at a loss for any long period.

Table 7 compares essential data on Indiana coal-mine accidents for the years 1930 and 1931. The comparison is not favorable for 1931. Although the explosion that occurred in 1931 is accountable in part for this unfavorable comparison, yet if the 28 fatalities due to the explosion are ignored, the comparison is still unfavorable.

Table 5.- Compensable accidents by causes for the year October 1, 1929, to September 30, 1930

Cause	Accidents Number	Accidents Per cent	Days lost Number	Days lost Per cent	Compensation paid Amount	Compensation paid Per cent
Underground:						
1. Falls of rock	86	13.7	56,364	44.1	$61,353.30	32.1
2. Falls of coal	74	11.8	9,918	7.8	17,687.98	9.2
3. Lifting coal or rock	36	5.7	1,351	1.1	3,632.44	1.9
4. Dropping coal or rock	7	1.1	192	.2	404.14	.2
5. Flying coal in eyes	29	4.6	925	.7	2,244.11	1.2
6. Falling objects	19	3.0	974	.8	2,543.06	1.3
7. Animals	9	1.4	434	.3	727.60	.4
8. Stumbling	18	2.8	2,709	2.1	6,147.07	3.2
9. Caught between car and rib or timber	33	5.2	4,191	3.3	10,861.93	5.7
10. Caught between car and face	9	1.4	6,407	5.0	6,374.95	3.3
11. Run over by cars	21	3.3	2,276	1.8	7,947.39	4.2
12. Pushing cars	36	5.7	1,513	1.2	3,240.55	1.7
13. Lifting cars	14	2.2	1,190	.9	2,618.65	1.5
14. Caught between cars	39	6.2	9,579	7.5	13,792.90	7.2
15. Miscellaneous cars	15	2.4	716	.6	1,555.29	.8
16. Caught between car and locomotive	13	2.1	7,366	5.8	11,565.49	6.1
17. Caught between locomotive and roof	4	.7	177	.1	390.79	.2
18. Caught between locomotive and rib or timber	-	-	-	-	-	-
19. Run over by locomotive	2	.3	651	.5	1,977.00	1.0
20. Gas and dust explosion	3	.5	6,236	4.9	5,497.46	2.9
21. Electricity (trolley)	3	.5	68	.05	122.57	.07
22. Electricity (other circuit)	5	.8	1,140	.9	1,025.14	.5
23. Mining machines	27	4.3	1,829	1.4	6,450.25	3.4
24. Mining machine jack pipe	13	2.1	289	.2	1,376.88	.7
25. Loading machine and conveyors	7	1.1	6,195	4.9	6,060.27	3.2
26. Other machinery	6	.9	192	.2	361.50	.02
27. Explosives	4	.7	888	.6	4,198.57	2.2
28. Tools	26	4.1	844	.6	1,605.86	.9
29. Miscellaneous	32	5.1	849	.6	1,489.13	.8
Shaft:						
30. Falling down	1	.2	112	.09	2,447.00	1.3
31. Material falling	-	-	-	-	-	-
32. Miscellaneous	-	-	-	-	-	
33. Cage	2	.3	65	.05	1,371.00	.7
Surface:						
34. Railroad cars and locomotives	4	.7	264	.25	658.31	.3
35. Bursting boiler or steam pipe	1	.2	21	.01	33.00	.01
36. Machinery	8	1.3	142	.15	226.63	.1
37. Electricity	-	-	-	-	-	-
38. Falling objects	12	1.8	888	.7	1,870.74	1.0
39. Miscellaneous	12	1.8	704	.6	1,398.71	.7
Total	629	100.0	127,659	100.0	191,257.66	100.0

Table 6.- Compensable accidents by causes for year October 1, 1930, to September 30, 1931

Cause	Accidents		Days lost		Compensation paid	
	Number	Per cent	Number	Per cent	Amount	Per cent
Underground:						
1. Falls of rock	117	10.5	47,478	13.9	$59,555.44	15.9
2. Falls of coal	110	10.0	22,742	6.6	24,855.50	6.7
3. Lifting coal or rock	66	6.0	4,480	1.3	10,694.61	2.9
4. Dropping coal or rock	51	4.6	2,019	.6	4,910.62	1.3
5. Flying coal in eyes	40	3.6	2,436	.7	6,110.48	1.6
6. Falling objects	37	3.3	1,297	.3	3,904.37	1.1
7. Animals	24	2.2	7,180	2.1	5,513.75	1.5
8. Stumbling	46	4.1	2,932	.9	7,490.45	2.0
9. Caught between car and rib or timber	45	4.0	2,451	.7	5,932.12	1.6
10. Caught between car and face	4	.4	148	.04	283.93	.08
11. Run over by cars	32	2.9	8,384	2.5	10,226.94	2.7
12. Pushing cars	53	4.8	1,958	.6	3,910.63	1.1
13. Lifting cars	40	3.6	13,247	3.8	11,825.17	3.2
14. Caught between cars	52	4.7	2,359	.7	5,810.15	1.5
15. Miscellaneous cars	39	3.5	3,803	1.1	8,915.77	2.4
16. Caught between car and locomotive	16	1.4	7,191	2.1	9,173.03	2.5
17. Caught between locomotive and roof	4	.4	61	.01	77.79	.02
18. Caught between locomotive and rib or timber	13	1.2	708	.3	1,395.10	.3
19. Run over by locomotive	6	.5	577	.2	1,749.15	.5
20. Gas and dust explosion	42	3.7	174,445	51.0	149,382.56	40.1
21. Electricity (trolley)	2	.2	6,035	1.8	5,121.00	1.3
22. Electricity (other circuits)	16	1.4	6,284	1.8	5,491.28	1.5
23. Mining machines	43	3.8	4,290	1.2	1,453.63	.4
24. Mining machine jack pipe	9	.8	257	.07	441.09	.1
25. Loading machine and conveyors	44	3.9	7,589	2.2	11,291.86	3.0
26. Other machinery	12	1.1	719	.2	1,506.21	.4
27. Explosives	-	-	-	-	-	-
28. Tools	37	3.3	1,019	.3	1,928.31	.5
29. Miscellaneous underground	38	3.4	1,288	.4	2,797.32	.8
Shaft:						
30. Falling down	2	.2	6,000	1.75	5,050.00	1.4
31. Material falling	1	.1	144	.04	322.93	.09
32. Miscellaneous	3	.3	164	.04	337.00	.09
33. Cage	4	.4	176	.05	400.62	11
Surface:						
34. Railroad cars and locomotives	13	1.2	391	.1	722.70	.2
35. Bursting boiler or steam pipe	-	-	-	-	-	-
36. Machinery	8	.7	340	.09	1,219.80	.3
37. Electricity	1	.1	28	.01	39.60	.01
38. Falling objects	14	1.2	538	.2	965.63	.2
39. Miscellaneous	29	2.5	1,128	.3	2,342.63	6
Total	1,113	100.0	342,186	100.0	372,249.17	100.0

Table 7.- Essential data on coal-mine accidents in Indiana during the fiscal years 1930 and 1931

Item	1930	1931	Increase or decrease: Number	Increase or decrease: Per cent
Average employees	10,698	9,948	-750	7
Coal mined, tons	10,887,499	9,056,810	-1,830,689	16.8
Man-days worked	1,728,974	1,440,005	-288,969	16.7
Man-hours worked	13,831,792	11,520,040	-2,311,752	16.7
Fatalities	23	65	+42	182.6
Lost-time accidents	2,303	2,248	-55	2.3
Compensable accidents	629	1,113	+484	76.9
Fatal accidents per million man-hours of exposure (United States average 1.8)	1.66	5.64	+3.98	239.7
Lost-time accidents per million man-hours of exposure	166.5	195.1	+28.6	17.1
Compensable accidents per million man-hours of exposure	46.8	96.6	+49.8	106.4
Days lost from compensable accidents	127,659	342,186	+214,527	168
Compensation paid	$191,258	$372,249	+$180,992	94.5
Per cent of total employees receiving lost-time injuries	21.5	22.5	-	-
Per cent of total employees receiving compensable injuries	5.8	11.1	-	-
Days lost per compensable accident	54	152	+98	181
Coal mined per fatality, tons	473,369	139,335	334,034	70
Fatalities per million tons (United States average 1911-1928, 3.77)	2.11	7.16	+5.05	239
Fatalities per 1000 men employed (United States average 1911-1928, 3.08)	2.14	6.53	+4.39	205
Coal mined per lost-time accident, tons	4,607	4,029	-578	12.5
Coal mined per compensable accident, tons	17,309	8,139	-9,170	53
Average sum paid per compensable accident	$304	$334	+$30	9
Premium cost per lost-time accident	$320	$262	-	-
Average time lost per compensable accident, days	203	307	+104	51
Cost of compensation insurance, cents per ton	6.7	6.4	-	-
Frequency rate (59.12 is average of 92 bituminous mines in National Safety Competition of 1931)	170	195	-	-
Severity rate (compensable accidents only)	9.2	30.2	-	-

I. C. 6674

December, 1932

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

SHAFT-SINKING METHODS AND COSTS, AND COST OF PLANT AND EQUIPMENT AT THE MACASSA MINE, KIRKLAND LAKE, ONTARIO

BY

G. A. Howes and Chas. F. Jackson

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SHAFT-SINKING METHODS AND COSTS, AND COST OF PLANT AND EQUIPMENT AT THE MACASSA MINE, KIRKLAND LAKE, ONTARIO[1]

By G. A. Howes[2] and Chas. F. Jackson[3]

INTRODUCTION

This is one of a series of circulars dealing with the cost of equipping and developing mining properties in the United States and Canada, prepared and published by the Bureau of Mines in cooperation with various mining and exploration companies. This group of circulars is designed to supplement an earlier series which covered the methods and costs of mining and milling at a large number of producing properties.

ACKNOWLEDGMENTS

The authors desire to thank Robert A. Bryce, president of Macassa Mines (Ltd.), for permission to use the data presented in this paper, and to acknowledge the cooperation of A. J. Keast, under whose direction as manager the plant was installed and equipped and the shaft was sunk to a depth of approximately 2,000 feet.

PURPOSE OF SHAFT

The shaft was sunk to develop the Macassa property along the extension of the main Kirkland Lake fault zone to the west of the active producing properties in the Kirkland Lake district.

Geological indications, together with the desirability of having an operating shaft centrally located on the property, favored sinking the shaft 2,500 feet west of the eastern boundary. The property of the Kirkland Lake Gold Mines (Ltd.) adjoins the Macassa on the east and is the deepest mine in the district, being developed to a depth of about 4,900 feet.

During the sinking of the Macassa shaft a drift was driven from the 2,475-foot level of the Kirkland Lake Gold mine westward along the ore zone to intersect the new shaft at a depth of approximately 2,500 feet. The sinking and drifting were timed so that shaft and drift would reach the objective point at the same time and connect. When this connection was made there was thus a vertical block of ground approximately 2,500 feet long by 2,500 feet high which could be rapidly explored and developed from the shaft and drift.

GEOLOGICAL FORMATIONS

The shaft was sunk through shallow overburden to solid rock. It then passed through 650 feet of conglomerate, 1,040 feet of igneous rocks, and through conglomerate for the balance of the distance. The conglomerate was hard, especially in the lower part of the shaft,

1 - The U. S. Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6674."

2 - Superintendent Macassa Mines (Ltd.).

3 - Principal mining engineer, U. S. Bureau of Mines.

while the igneous rocks were easy to drill. The ground was for the most part firm, but in the igneous rocks excavation and timbering were followed promptly by close lagging of the sets. At one point in the shaft a heavy flow of water was encountered which required the injection of cement to hold it back.

PREPARATION PERIOD

Before the shaft site was definitely decided upon, considerable diamond drilling had been done to determine the geological formations and structure. The shaft was located within 200 feet of one of the drill holes and the character of the formations to be encountered was thus approximately known in advance of sinking operations.

The surface at this location was covered with second-growth spruce, poplar, and birch. An area of about 8 acres was cleared to prepare the site for the erection of mine buildings, for rock-dump room, and as a protection against fire. The wood cleared from this area was utilized for fuel during the winter months.

Materials, plant equipment, and supplies were hauled over 1 mile of improved government road and about a half mile of mine road. The cost of building the mine road and clearing the site was $2,297.

BUILDINGS

Proximity to the town of Kirkland Lake made it unnecessary to construct camps at the property. Ten small buildings were constructed to house the mine plant and office, as follows:

Building	Size, feet	Type of construction
Office	20 x 20	Frame: 2 x 4 inch joists, rafters, and studding. Covered with fir and good-quality roofing; finished inside with gyproc.
Hoist and compressor house	30 x 50	Do.
Dry house	19 x 25	Do.
Boiler house (for heating plant only)	25 x 17	2 x 4 inch spruce lumber; covered outside with good-quality roofing.
Blacksmith shop	17 x 22	Do.
Topman's house	11 x 11	Same as office.
Pump house	10 x 8	Same as boiler house.
Substation	12 x 12	Same as office.
Powder magazine	14 x 16	Same as boiler house with addition of a good jack-pine lumber floor.
Cap house	6 x 8	Same as pump house, but with good floor.

Cost of mine buildings

Office	$784
Dry house	787
Hoist and compressor house	1,960
Boiler house	380
Blacksmith shop	561
Topman's house	249
Pump house	97
Substation	558
Powder magazine	455
Cap house	23
Total mine buildings	$5,854

Although the structures listed were required in connection with shaft-sinking operations, they were also designed to serve during later underground exploration and development; thus the total costs are not strictly chargeable to shaft sinking. The same is true of the larger items of equipment cost which are listed in the next section of the paper.

SURFACE PLANT EQUIPMENT

Description.- The electrically driven hoist and compressor, transformers and substation equipment, steel sharpener and other shop equipment, surface pipe lines, pumps and heating system, and shaft headframe comprise the principal plant items.

The hoist is a 72 by 36 inch, double-drum, electric hoist having a capacity of 15,000 pounds on a single line at a speed of 1,000 feet per minute and is powered by a 150-hp., 550-volt, 3-phase, 25-cycle induction motor. It is equipped with two Lilly hoist controllers two brake regulators, position indicators, etc.

The compressor is a 20 - 12 by 14 inch angle-compound machine having a capacity of 1,000 cubic feet of free air per minute at 95 pounds.

The shaft house is a 6-post, A-type timber structure, 60 feet high and fully enclosed with painted corrugated-iron sheathing. There are two 6-foot head sheaves and two compartments equipped with the necessary safety doors and chutes for dumping the sinking buckets.

Substation equipment consists of two 11,000 to 575 volt, 25-cycle, 200-K.V.a., outside-type oil transformers with lightning arrestors, outside poles, and insulators and the usual inside switchboard equipment.

Surface Plant Equipment Costs

The following figures cover the costs of plant equipment, including freight and haulage and the cost of installation.

Equipment	Cost
Office equipment	$636
Hoist and installation except foundations	25,548
Hoist foundations	1,275
Headframe	2,938
Compressor and installation (except foundations)	7,649
Compressor foundations	845

Blacksmith shop equipment:	
Forge, small tools, etc.	337
Drill steel sharpener	2,462
Dry-house equipment (stove)	23
Cap house	75
Substation equipment	6,528
Water-supply system	327
Miscellaneous surface equipment	119
Surface track	57
Surface pipe lines	773
Surface pumps and motors	399
Heating system	827
Total surface plant equipment	$50,810

UNDERGROUND EQUIPMENT AND COST

Sinking equipment consisted of the usual rock drills, hose and accessories, drill steel, sinking buckets and crossheads, pumps, signaling system and small tools. The crossheads were of special construction and were designed to prevent the bucket from catching against the shaft timbers during hoisting. The construction is shown in Figure 1.

The cost of underground equipment is here itemized:

Equipment	Cost
Rock drills, drill columns, and accessories; including 5 new 3 1/2-inch drifter machines	$2,282
Drill steel (4 1/2 tons)	1,095
Underground pumps and motors	3,137
Two 1 1/2-ton mine cars (on rock dump)	453
Three 3,000-pound sinking buckets	300
Total	$7,267

Note: The crossheads, blasting set, hoisting signals, mine tools, and drill steel were written into the operating costs and are included in the sinking costs, Table 1.

Summary of Preparation, Plant, and Equipment Costs

Preparation, plant, and equipment costs are summarized as follows:

Mine road and clearing site	$2,297
Buildings (not including headframe)	5,854
Plant equipment including headframe	50,810
Underground equipment	7,267
Total	$66,228

SHAFT SINKING

Actual shaft sinking was begun on May 25, 1931, and was completed on August 9, 1932. Shaft stations were cut at 250-foot intervals, and the total depth was 2,489 feet. During January, 1932, an advance of 205 feet was made.

The shaft is 9 by 17 feet,* rock dimensions, and is timbered with 8 by 8 inch framed B. C. fir timber sets, 16 feet 3 inches by 6 feet 9 inches over-all dimensions. It is divided

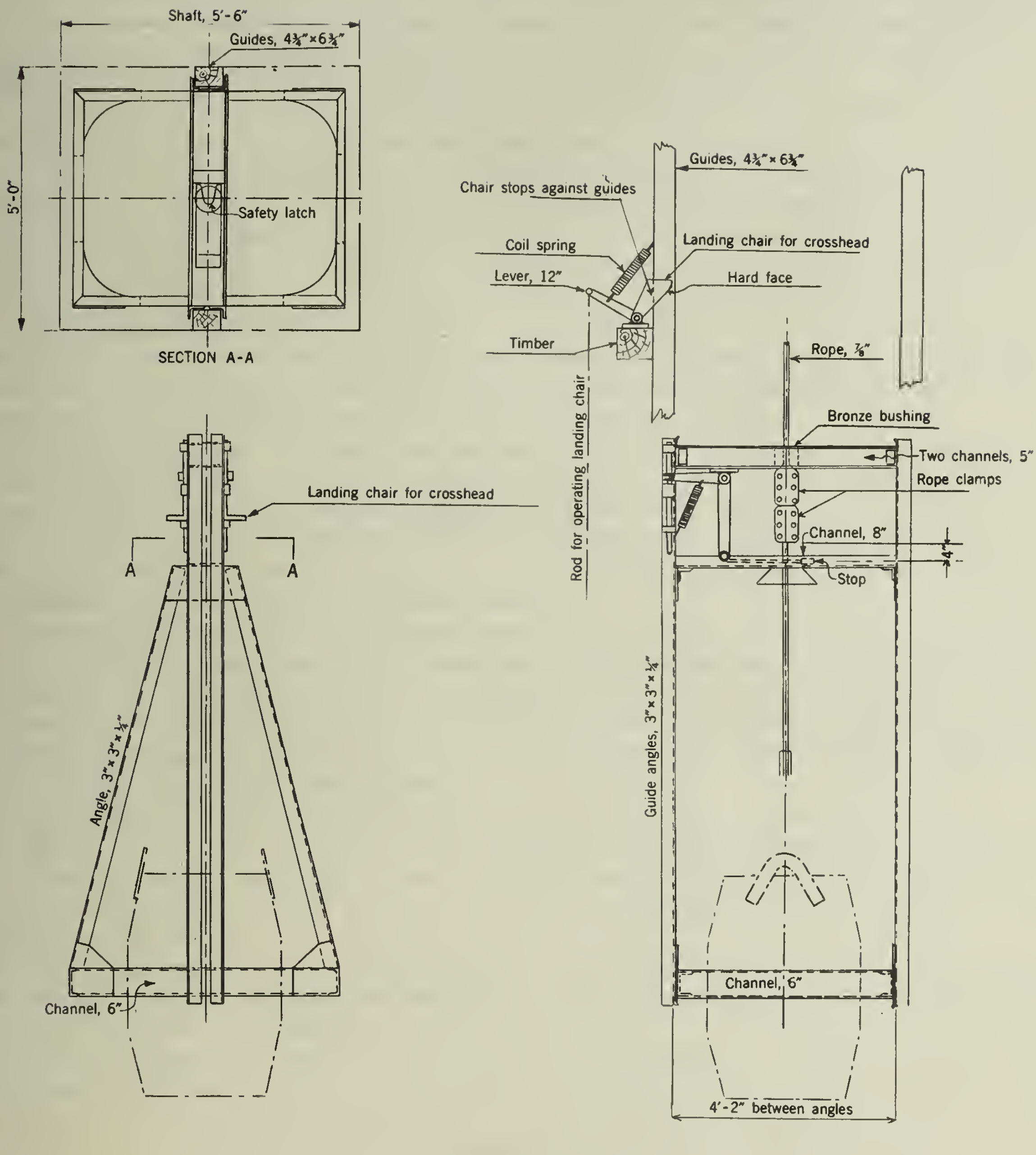

Figure 1.—Special crosshead

into two hoisting compartments each 5 feet 6 inches by 5 feet in the clear, and one pipe and manway compartment 5 feet 6 inches by 3 feet 9 inches, as shown in Figure 2. Sets are spaced on 7-foot centers and bearing timbers are installed 250 feet apart, or immediately below the level stations. The manway is separated from the hoisting compartments by 1 by 7 inch sheathing and the outside lagging is of 8-inch round spruce poles.

The manway compartment has landings at every 14 feet. The ladders are 16 feet long and are made of 2 by 4 inch spruce with 1 by 3 by 16 inch rungs.

Pipe lines consist of a 6-inch air line, a 4-inch water-discharge column, a $1\frac{1}{2}$-inch water line to supply the rock drills, and a 16-inch galvanized-iron ventilating pipe with stove-pipe joints. Tees were inserted in all pipe lines opposite the various levels (stations for which were cut at 250-foot intervals) to provide connections for later crosscutting and drifting operations.

Ventilation.- An exhaust fan driven by a $7\frac{1}{2}$-hp., 550-volt, a.-c. motor was installed at the shaft collar to draw smoke through the 16-inch galvanized-iron pipe. The fan was turned on after each blast, and the men returned to work within one-half hour after blasting. The time thus saved soon repaid the initial cost of the fan and pipe.

Sinking Practice.- The shaft was excavated by hand to bedrock, which was within 2 feet of the surface at one end of the shaft and within 10 feet at the other end. A small air-driven tugger hoist was then used to sink to a depth of 40 feet below surface prior to building a concrete collar and erecting the regular headframe, and while the regular hoist was being installed. The concrete collar is 15 feet high, extending from solid bedrock to approximately 5 feet above the original surface.

After installation of the hoist and headframe, sinking was carried on regularly on three shifts of 8 hours each. Each shift consisted of six miners, one of whom acted as shift leader, and all of whom worked under the direction of the mine superintendent. A deckman was employed at the shaft collar on each shift and during mucking and timbering periods was assisted by a helper in tramming the rock to the dump and in handling timber. The deckman also cared for the dry and the heating plant.

A topman was employed regularly on day shift but was on call at all hours of the day and night. His duties included rock-drill repairing, pumping, and general utility work.

One blacksmith sharpened the drill steel and performed general blacksmithing work. When there was heavy work to be done - on the average about one day per week - a helper was engaged by the day.

One hoistman on each of the three shifts operated the hoist and looked after the air compressor. One of the hoistmen ranked as a boss and was responsible for the care of the hoist and compressor equipment.

An accountant had charge of the stores, timekeeping, and other office work, and kept the company accounts.

Cycle of Operations

As already stated the shaft crew was composed entirely of miners; that is to say, of men who were competent to perform any work in the shaft. Each shift therefore simply carried on the work from the point at which the preceeding shift left off, whether that work was drilling, mucking, or timbering. Drilling and mucking usually proceeded steadily for five to seven days, or until the bottom of the shaft was 35 to 40 feet below the last set of timber. The shaft was then timbered to within about 10 feet of the bottom.

Drilling.- The type of round found to be the most efficient in breaking the ground is shown in Figure 3. An average of 38 holes was drilled per round, 16 of which were cut holes and 22 of which were square-up holes. With six men in the shaft using four drills, the

average drilling time, including time consumed in taking down drills, gear, and steel, was 4½ hours. Five drill machines were taken down in order to provide a spare machine in case of breakdown. These machines were 3½-inch, water-Leyner type drifters, equipped with plugger handles. After each drilling period the machines were sent to the surface where they were carefully inspected, overhauled, and oiled by the topman. Any broken parts were immediately replaced and the machines put in first class shape for use in drilling the next round.

Drill steel was of 1-inch, quarter-octagon section with standard cross bits. Four 2-foot steel changes per hole were used to a depth of 2,000 feet, at which depth the rock became so hard as to require five changes. The changes and bit gages were as follows:

Changes	0 to 2,000 feet		Below 2,000 feet	
	Length	Gage	Length	Gage
Starters	2'-6"	1-7/8"	2'-0"	1-7/8"
Seconds..	4'-6"	1-3/4"	3'-6"	1-13/16"
Thirds....	6'-6"	1-5/8"	5'-0"	1-3/4"
Fourths..	8'-6"	1-1/2"	6'-6"	1-11/16"
Fifths....	-	-	8'-6"	1-5/8"

An average of 195 pieces of drill steel was used per round. Drills, steel, scaling bars, and tools were lowered and hoisted in one of the sinking buckets.

Wet drilling was practiced throughout the operation. Air pressure at the machines was maintained as nearly as possible at 85 to 90 pounds. Air and water were supplied to the rock drills through a header or manifold. Between the drilling periods this header was left at the shaft station nearest to the bottom of the shaft but when required for drilling it was lowered on the end of one of the hoisting cables to a convenient height above the shaft bottom.

One hoisting compartment was used for bailing water during the drilling shift, and the other hoisting compartment contained the header.

Blasting.- Both 40 per cent and 50 per cent gelatin dynamite were employed in 1-1/8 by 8 inch cartridges. Fifty per cent dynamite was used in the cut holes but only occasionally in the square-up, 40 per cent strength being usually employed in the latter. The miners judged the nature of the ground from the way it drilled and thus determined whether 50 per cent explosive was necessary or not in the square-ups.

Delay-action, all-metal detonators (0 to 10 delays) were used and were fired from an electric switch in the shaft house. A cab-tire cable was run from this switch to the bottom of the shaft, additional cable being added as the shaft was deepened. The detonators were connected in series by No. 20 insulated connecting wires. The detonators were placed in a priming cartridge halfway down each hole. They were entirely waterproof and no trouble was experienced with short circuits or misfires. The blasting circuit carried 30 amperes at 125 volts and was closed by a switch having a special connector which could only be inserted after the switch box was unlocked. Each shift leader carried a key to this box and was alone responsible for the blasting. The order of firing is shown in Figure 3, where the circled numbers indicate the number of the delay used in each hole.

A 2-ton blasting set was kept chained to the bottom of the last set of shaft timbers to protect it from flying rock during blasting. This was made of 8-inch channel iron cut and riveted together to match the shaft timbers in plan. During the timbering periods the blasting set was lowered for the required distance with 1½-ton chain blocks, two at one end of the

set and the hoisting cable at the other.

Mucking.- Down to a depth of 1,900 feet the broken rock was shoveled into barrel-shaped buckets having a capacity of 2,200 pounds each. Three buckets were used, one being at the bottom of the shaft while the other two, one loaded and the other empty, were being hoisted and lowered in balance in the shaft.

Upon reaching a depth of 1,900 feet the shaftmen were delayed while waiting for the empty bucket to be returned. It was therefore decided to increase the bucket load to 3,000 pounds, and three 3,000-pound buckets, 4 feet high and having vertical sides, were placed in service.

Buckets were attached to the hoist cables with strong clevices made from 3 by 1/2 inch mild steel and reinforced with 1/4-inch steel plate welded on the inside, the hole being bored through both. A 1-1/2-inch chrome-steel bolt with a large hexagon nut held the bucket.

As previously stated, six men did the mucking. The cut required from 3 to $3\frac{1}{2}$ hours to muck and contained from 14 to 20 buckets. The square-up was mucked in from 5 to 7 hours and contained from 41 to 54 buckets.

Timbering.- Usually four sets of timber are put in during each timbering period. The blasting set is first lowered far enough to make room for a set; the wall plates are then lowered in a bucket and hung with 3/4-inch, round-iron hanging rods. The end plates, dividers, and posts are lowered, placed, and aligned and the timbers are securely blocked in place with 8 by 8 inch blocking and sawed wedges. Round spruce lagging poles 8 inches in diameter were placed outside the timbers and spaced as required by the nature of the ground in each instance. The average time required to place a set of timber was 2 hours and 30 minutes. The manway and hoisting compartments were lagged off with a solid lining of 1 by 7 inch rough spruce boards. Shaft timbers were framed for the entire shaft by a man working on contract during the early stages of shaft sinking.

Cementing-off Water.- At a depth of about 400 feet a rather heavy flow of water was encountered in the shaft. It entered through a series of narrow fractures in the rock and considerably slowed up shaft-sinking progress. To overcome this, holes were drilled into the shaft walls across the fractures and cement was forced into the holes with a No. 7 sinking pump. This practically stopped the flow of water into the shaft.

Pumping.- Water in the shaft came from the fractures previously mentioned, from the surface, and from the rock drills. The water from drilling was baled from the shaft during the drilling shift. Cement rings were installed immediately above the 500 and 1,000 foot levels, with pipes leading to sumps at each of these levels.

The pump installed on the 1,000-foot level had a capacity of 110 gallons per minute and was only operated about one hour per day on the average. It pumped against a total head of 1,020 feet and was installed in a permanent manner for rehandling water from the bottom levels during later development work.

At the 2,000-foot level a 50-gallon per minute pump, operating against an 1,100-foot head, was installed to relay water from the sump at the bottom of the shaft. This was driven by a 20-hp. motor and pumped to the sump on the 1,000-foot level. A 50-gallon pump, driven by a 10-hp. motor, handles the water from the bottom to the sump on the 2,000 level.

During sinking operations, the 4-inch discharge column was carried down as the shaft was deepened, a length of pipe being added each time timbering was done.

The permanent power cable was installed at 500-foot intervals during sinking, as were the electric signal wires. Until the 500-foot level station and sump had been cut, water was handled entirely by bailing, and hoisting signals were transmitted by an ordinary bell cord to a bell in the hoist room. The power cable is a 3-conductor, No. 8, B. and S., galvanized steel-wire-armored cable. The signal wires were No. 5, 7-conductor cable.

The power cable was secured to the shaft timbers by means of 3/8 by $1\frac{1}{2}$ inch steel clips and the signal cables by 1/8 by 1 inch clips. Both of these cables were in 540-foot lengths

and were lowered into the shaft on the reels. They were then hoisted up the manway compartment by means of the hoist, the hoisting cable having been swung over into that compartment.

COST OF SINKING

The sinking costs per foot are shown in Table 1, in which the total cost is distributed over the several operations involved. The costs given include part of the cost of a number of items of equipment such as drill steel, sinking buckets, hoisting cable, etc., which have been absorbed into the operating account.

Development costs, whether they be for shaft sinking, drifting, or crosscutting and raising, are usually considerably higher during work purely of a development nature than are similar costs at a producing property, since development work must absorb all the pumping, surface costs, and overhead charges which in the latter instance are borne in large part by ore production. The figures in Table 1 include all such charges.

Table 1.- Shaft-sinking costs per foot sunk (2,489 ft.)

	Sinking labor and supervision	Drill, drill repairs, steel, air and water lines	Power	Explosives	Timber	Concrete collar	Other supplies	Total
Drilling and blasting	$10.378	$3.564 (mat'l) 3.062 (labor)	$2.899	$6.367	-	-	$0.218	$26.488
Mucking	10.884	-	-	-	-	-	.166	11.050
Timbering	3.546	-	-	-	6.087	-	-	9.633
Concreting	-	-	-	-	-	0.215	-	.215
Hoisting	2.987	-	2.971	-	-	-	2.224	8.182
Decking and rock disposal	3.570	-	-	-	-	-	.033	3.603
Ventilation	-	1.519	.050	-	-	-	-	1.569
Pumping	-	-	.536	-	-	-	1.698	2.234
Supervision and workmen's compensation	4.810	-	-	-	-	-	-	4.810
Miscellaneous	-	-	.125	-	-	-	.509	.634
Total direct costs	36.175	8.145	6.581	6.367	6.087	0.215	4.848	68.418
Proportion of general charges								2.512
Total cost per foot sunk								$70.930

The drift driven from the adjoining Kirkland Lake Gold Mines (Ltd.) property was driven by and under the direct supervision of the staff of that company on contract by special arrangement, so that all supervision charges were carried by the shaft-sinking operation.

The scale of wages paid was as follows:

Hoistmen	$5.20, plus bonus depending upon footage.
Deckmen	4.15 Do.
Deckmen helpers	.40 per hour.
Topman	$150.00 per month, plus 20 cent bonus for footage over 150 feet.
Blacksmith	7.00
Surface laborers	.40 per hour.
Shaftmen	6.00, plus bonus.

The bonus scale for shaftmen was as follows:

Footage per month	Wage plus bonus
140	$6.53
160	7.33
180	8.40
200	9.73
210	11.06
220	12.40

The shaftmen paid for their own shovels in excess of a consumption of 12 per month and for all explosives over $5.60 per foot. They earned an average of about $8.50 per shift.

Table 2 gives the costs in units of labor, power, and supplies.

Unit labor costs are given for the month of April, 1932, a typical month, when 171 feet of sinking and a shaft station were completed. Costs do not include the cutting of the station. Other unit costs cover the entire sinking period.

Table 2.- Costs in units of labor, power, and supplies

April, 1932. 171 feet sunk.

Labor:	Man-hours per foot
Drilling, blasting, and blowing smoke	8.52
Mucking	11.53
Timbering	2.61
Bailing water	.65
Miscellaneous and delays	.93
Total shaft crew	24.24
Feet per 8-hour man-shift - shaft crew only,	0.33
Surface crew:	Man-hours per foot
Hoistmen	4.44
Dumping and disposal of rock	7.90
Blacksmithing, steel sharpening, drill repairs and general surface	3.28
Supervision and clerical	2.80
Total man-hours per foot surface	18.42
Feet per man-shift, surface	.433
Total crew-shaft and surface:	
Man-hours per foot	42.66
Feet per 8-hour man-shift	.187

Table 2.- Costs in units of labor, power, and supplies - Continued

Materials and supplies (2,489 feet of shaft):	Per foot of shaft
Explosives:	
Strength	Pounds
40 per cent	19.0
50 per cent	7.7
60 per cent	2.5
Total all strengths	29.2
Timber:	
	Board measure
Shaft sets (8 x 8 inch timber)	85.0
Blocking (8 x 8 inches)	17.2
Sheathing (1 x 7 inches)	5.0
Guides	12.5
Total	119.7
	Linear feet
Round lagging (8 inch poles)	18.0
	Pounds
Drill steel	2.0

Table 3.- Performance Data

Time worked in shaft (2,489 feet sunk)	days	412
Average progress per day of 24 hours	feet	6.04
Progress per round	Do.	4.93
Number of rounds completed		505
Average rate of sinking plus 18 feet of station and 15 feet of crosscutting per month	feet per month	170
Drilling:		
Number of holes per round		38
Number of steels sharpened per round		195
Total drilling per round	feet	300
Drilling per foot of advance	Do.	59.64
Average number of buckets hoisted per round		67.7
Average number of buckets hoisted per foot of advance		13.45

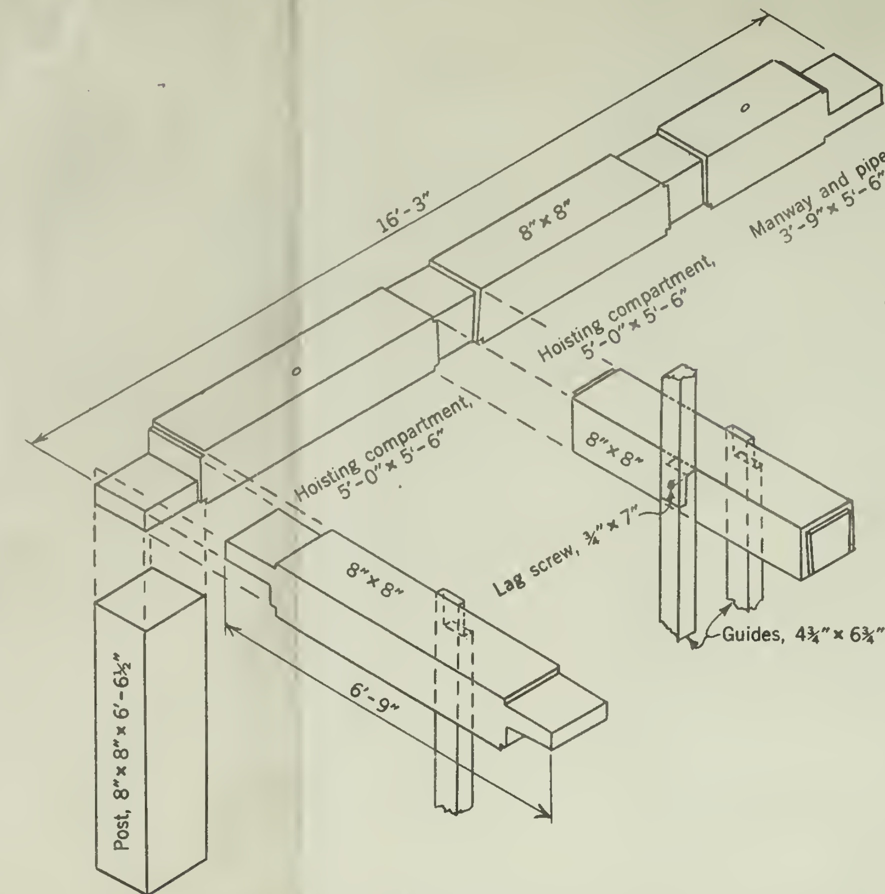

Figure 2.—Shaft timbering

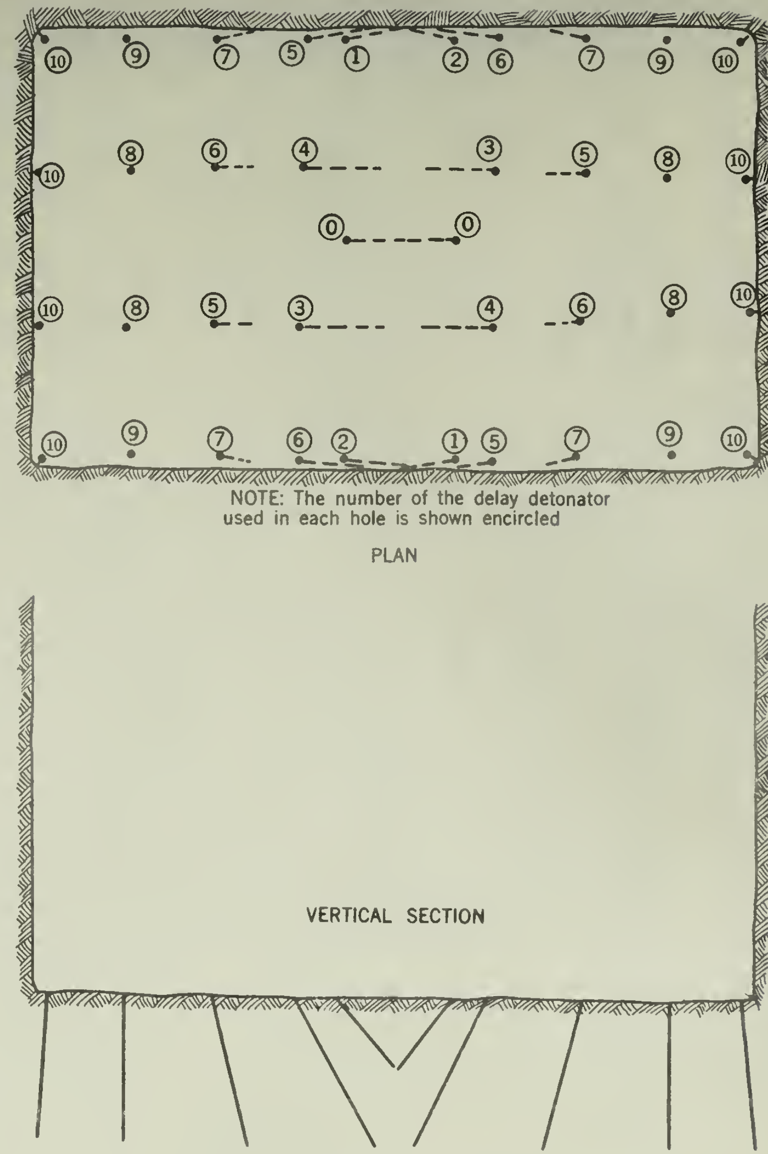

Figure 3.—Shaft round

I.C.6675,
December, 1932.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

SAFETY PRACTICES AT MINE 1, SPRING CANYON COAL CO., UTAH[1]

By D. J. Parker[2]

The oft-repeated expression that the taproot of safety lies in the attitude of the management toward accident-prevention work is indisputable and has become almost an axiom. Trite and somewhat timeworn though the statement may be, it is nevertheless true that the supervisory force must earnestly support a policy of safety and maintain at all times a considerate regard for the welfare of the employee if that high degree of cooperation of the employee, so vital to the successful prosecution of any safety program, is to be expected.

The responsibility for preventing accidents rests upon the management, and in the case of the Spring Canyon Coal Co. such responsibility is recognized and accepted with little or no reservation. This company, aside from the humanitarian aspects involved, believes that safety is good business and not a matter to be treated passively or in a perfunctory manner.

The problems with respect to safety that have been met and wholly or partly solved can best be appreciated when some of the natural conditions, adverse and otherwise, existing at this operation are taken into consideration.

Mine 1 is about 1 mile north of the town of Spring Canyon, Carbon County, Utah, and is served by both the Denver & Rio Grande Western and Utah Coal Route railways. During the year 1931 the mine produced 312,397 tons, approximately 10 per cent being derived from pillar workings. The normal daily output during the summer months is about 2,000 tons, and during the peak of production in the winter season the daily production varies from 2,500 to 3,000 tons.

The number of miners varies from 87 in summer to 220 in winter, and the day men vary in number from 66 in the winter to 45 in the summer; 12 company men are employed the year around.

The officers of the company are J. B. Smith, president, San Francisco, California; T. R. Stockett, general manager, Salt Lake City; and G. A. Murphy, superintendent, David Brown, assistant superintendent, and John Sullivan, mine foreman, all of Spring Canyon, Utah.

COAL BED

The mine is opened on the bottom sub-bed, which averages about 8 1/4 feet in thickness but at times attains a maximum of 9 feet. The pitch of the bed is in a northeasterly direction with a dip of 8 to 10 per cent. It occurs in the Mesaverde formation of the Upper Cretaceous system and is one of the important beds of the Book Cliffs field.

The bottom consists of massive sandstone and has no tendency to heave. The character and quality of the roof material vary considerably; in the upper levels which are now worked out for the most part, the roof is excellent consisting of about 3 feet of blue slate; in the lower sections of the mine the immediate roof consists of two 7-inch layers of cap rock and lying above the cap rock is a laminated sandstone which is treacherous and breaks to a considerable height when unsupported.

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6675."

2 - District engineer, U. S. Bureau of Mines Safety Station, Salt Lake City, Utah.

The thickness of the overburden varies from a few hundred feet near the outcrop to approximately 3,000 feet over the most advanced workings. As a result of the heavy cover, bounces are not uncommon; they occur principally in the pillar sections, and as their occurrence can not be predicted, pillar extraction became so extremely hazardous that it was discontinued in certain sections of the mine.

A well-constructed steel tipple of modern design serves the operation. It is equipped with shaking and vibrating screens, rotary dump, reciprocating feeder, crusher, loading booms, car retarders, three box-car loaders, rescreening plant, and anthracite-type spiral-gravity separators. Twenty different sizes of coal can be prepared, any six of which can be made simultaneously. The mechanical arrangement is such that the change from one size to another can be effected without interrupting plant operation.

Mechanical safeguards are provided wherever needed, and ample fire protection is afforded. Hazards due to coal-dust accumulations in such plants are fully recognized and measures are taken to guard against them.

Much attention has been paid to proper wiring and safeguarding, as far as possible, of all existing hazards incidental to the safe operation of the surface plant. Good housekeeping was in evidence in the maintenance of shops, tipple, magazines, substation, fan and lamp houses, and yards; such orderliness is an important phase of any safety program.

DEVELOPMENT

The mine is opened by parallel slopes which are now approximately 2 miles long. An additional slope and air course, known as the west slope, parallels the main slopes 3,100 feet to the west. The room-and-pillar system of mining is employed, and essentially level entries and air courses are turned to the right and left off the main slope, the intent being to have a slight grade in favor of loaded cars on haulage roads.

Rooms are turned at an angle of 45° off the entries only and are driven 22 feet wide and 350 feet deep. The distance between room centers, measured along the entry, is 100 feet. Slopes, entries, and air courses are driven 14 feet wide on 75-foot centers. The distance between centers of crosscuts in rooms and on entries is 75 and 200 feet, respectively.

The character of the roof material and the depth of overburden were important factors in determining the sizes of pillars maintained, both as to safety and economy.

UNDERCUTTING, LOADING, GATHERING

Prior to blasting, the coal is undercut with shortwall machines, eight of the twelve being of permissible type. Electric drills of both permissible and nonpermissible types are employed; it is planned to replace all nonpermissible face equipment with permissible equipment.

All loading is accomplished by hand. Mechanical loading does not lend itself to economical operation of the mine, due to adverse roof conditions and to the necessity for cleaning the product as far as possible at the face.

The coal is gathered by trolley-reel and combination trolley-storage-battery locomotives and delivered to the several slope partings by main-line trolley locomotives. The main slope hoist delivers the coal to a side track near the top of the slope, from which point it is hauled to the winding engine yards about 500 feet inby the mine portal. The winding engine drops the coal finally to the tipple over a 3,700-foot incline.

HAULAGE

The importance of safety in connection with transportation has not been overlooked. Some of the more notable features in this connection may be summarized as follows: (1) Ample

clearance and shelter holes, (2) guarded trolley wires where necessary, (3) track and rolling stock maintained in excellent condition, (4) all cars equipped with safety chains, (5) a rigid daily inspection of haulage equipment, (6) heat treatment of pins and links at least every six months, (7) periodic inspection of hoisting ropes, (8) red lights carried on the rear end of all underground trips, (9) proper blocking of cars at the face, (10) flying switches prohibited, and (11) trips pulled instead of pushed.

Derail on Outside Incline

Near the foot of the steepest part of the outside incline a derail switch is maintained. It is the sole duty of one man to keep this derail open at all times, except when trips are passing over it. The operator is housed in a tower near the track and has an unobstructed view of the incline at all times; it would be feasible to operate the derail automatically, but the management, through foresight and a paramount interest in safety, chose the more expensive but safer and more dependable method of assigning a man to the job. An automatically operated derail is always likely to become blocked with snow, coal, or other debris so as to cause wrecks, with all their attendant costs and dangers.

A magnetic derail switch is in the right-hand rail on the main slope track at 10 left entry; two electromagnets, one for opening and the other for closing the derail, are used and operate on 250 volts d.c. Two bare wires, carrying 250 volts d.c., extend along the roof of the slope and to the right of the track; these wires which are exposed for a distance of about 50 feet, are about 250 feet above the derail, and there is a similar layout 250 feet below the derail. This circuit is on the opposite side of the slope from the regular signaling and telephone circuits.

To operate the derail, the trip rider, without any slowing down of the trip in descent, short-circuits the pair of wires above the derail. The derail closes instantly and stays closed until the trip rider opens it by short-circuiting the two bare wires below the derail. The derail is spring-connected to the throw, and ascending trips trail it successfully.

The trip rider uses a T-shaped copper conductor with an insulated handle in making the contact between the wires to bring about the short-circuiting mentioned. If for any reason the trip rider should fail to complete the circuit operating the derail, he would still have time to signal the hoistman to stop the trip before it reached the derail. Again, should the power "go off" the circuit controlling the derail while the trip is descending, the hoistman is signaled automatically.

For a distance of 150 feet below the derail the floor on the slope is covered with about 6 inches of incombustible material, consisting principally of adobe, sand, and rock dust; this is for the purpose of minimizing the possible explosibility of the dust cloud in case of the derailment of a loaded trip.

Sixteen-car trips are operated on the incline and on the main slope, and 8-car trips are hauled on the west slope. The main trips are attached to the rope in the mine between the winding engine and the portal; facilities are afforded for securely blocking both empty and loaded trips when they are delivered to the winding engine. No drags are employed either on the incline or the slopes. Red lights are used on the rear of all underground trips. Cars were found securely blocked at the faces; in rooms and entries it is the practice to set a tie or timber against the end of the car, with the lower end resting against a tie to which the rails are securely spiked.

Haulage on Main Slope

Cars at the face of the slopes are blocked by using a $\frac{1}{2}$-inch wire rope with a loop in both ends, the loops being made by splicing the ends into the rope. A loose tie is placed

under the rails several feet above the car and passes through one of the loops; a tie is similarly passed through the other loop and rests on top of the rails and against the lower pair of wheels.

The hoisting speed on the main slope is about 1,180 feet per minute, and for empty trips the speed is about 1,475 feet per minute. Ropes of 1 1/8 inches are used on the main and west slopes and 1½ inches on the incline. The ropes are of "sealed" construction, which means that the diameter of the outside wires is greater than that of the inside wires.

A clean, well-kept manway is maintained on the left side of the main slope. The manway portal is between the fan and the main slope hoist; it intersects the main slope about 150 feet inby the hoist, and for about 100 feet the left side of the slope is used as a manway. For this distance the manway is fenced off from the slope by wire rope securely fastened to a row of substantial timbers.

The manway, of necessity, crosses all of the left-hand entries (only a few of which are working) off the main slope. To protect men while crossing these entries, all switch stands on left turnouts from the slope are so wired that when the switch is opened two red lights show on the manway, one just above the entry and the other immediately below; the lights show until the switch on the slope is again closed.

No one is permitted on the slope except where duty requires. Man trips are operated both morning and afternoon on the main slope for the convenience of the men. Such trips are run at a restricted speed (about 600 feet per minute), and men are not permitted to get on or off the trips while they are in motion.

During the operation of hoisting and lowering men, an additional hoistman is on duty. This is a highly commendable practice.

TIMBERING

Not much timber is required in narrow work where the roof is good; where the roof is found to be unsound, however, legs and collars are used, and in some instances overhead cribbing is necessary. The rooms are systematically timbered; tracks are laid in the center of the rooms, and a row of props is carried on each side of the track and 30 inches from the rail; the distance between props, measured along the length of the room, varies from 5¼ to 6 feet. Normally, the timbering is kept within 12 feet of the face; however, temporary props or safety posts are placed much nearer the face wherever the necessity arises. In the pillar work the timbers are set much closer than in advancing rooms or entries. Cap pieces are set at right angles to the center of the room, except where slips are encountered; then they are placed at right angles to the slips. Cap pieces are hewn on two sides, and are about 24 inches long and 6 inches thick; they are made from essentially the same sized material as the props.

Break lines are carried on a 45° angle with the entry, and considerable effort is made to maintain them in a uniform manner.

Timber requirements appear to be well taken care of, as an adequate supply has been observed in all working places.

Picks are used for roof testing. Shot firers test the roof before blasting, and the fire bosses again test it while making their rounds prior to the arrival of the day shift. Machinemen also test the roof before setting up their equipment, and the miners make it a practice to sound the roof from time to time during the working shift. Since there is no mechanical loading in the mine, the opportunity for hearing any movement of the roof material is not reduced by the noise from the machinery at or near the places where most of the men work.

The mine is well timbered, and there is every indication that careful supervision and rigid discipline are exercised by the underground officials in this respect.

ELECTRIC POWER

Power enters the mine at 2,300 volts a.c. through heavily armored, submarine "pickproof" cables. The west slope section receives its power through a separate circuit; a 2,300-volt, armored submarine cable extends through a 603-foot borehole, which taps the workings on the west slope air course about 40 feet above 9 left entry, the borehole being cased with 2-inch galvanized pipe.

The winding engine is of tandem-drum type, and is operated by a 500-hp., 2,300-volt a.c. motor; it is equipped with herringbone gears, regenerative braking, air brakes, and an over-winding device. When the empty trips reach a certain point after landing in the mine, the power is automatically shut off; it is necessary for the hoistman to close a hand-operated switch before the current is again available.

A rubber mat is provided for the switch panel, and suitable fire extinguishers and rock-dust are available in case of fire. Bearings and brake drums are water cooled.

The energy developed by the regenerative braking feature is equivalent to 450 kilowatts.

The engine and hoist rooms are kept neat and clean. The walls of the hoist room are of concrete and the roof is of sandstone.

HOISTS

As previously stated, the winding engine is in the mine, and in addition to handling the output of Mine 1 on the incline, it also handles the output from Mine 3, farther up the canyon.

All gears and other moving parts of the engine and ropes are well guarded.

The water used to cool the bearings and brake drums is piped to an overhead sprinkler outby the winding engine for the purpose of wetting down empty trips upon their arrival from the tipple.

The main slope hoist is housed in a stone building on the surface, built over the portal of the main slope and directly against the face of the cliff; it is of the single-drum type, and is operated by a 500-hp., 2,300-volt a.c. motor. The switch panel has a rubber mat in front of it, and rock dust and suitable fire extinguishers are available in case of fire. Both this hoist and the winding engine are equipped with 220-volt a.c. control. All moving parts of the hoist are adequately guarded, and good housekeeping is in evidence in the hoist room.

Brattice boards, brattice cloth, tools, nails, and other similar materials are stored in the hoist room for emergency use.

The west slope hoist is of the single-drum type and is on the west slope about 475 feet above 9 left entry. Eight-car trips are hoisted from the lower parts of this side of the mine and dropped into 9 left entry, from which they are hauled by electric locomotives to the main slope. The hoist is operated by a 225-hp., 2,300-volt, a.c. motor.

The walls of the hoist room are of concrete, and the roof is supported by steel rails. The room is of thoroughly fireproof construction, and both rock dust and fire extinguishers are available in case of emergency.

Oil for the hoist is kept in a concrete vault provided with a steel door and recessed into the concrete wall of the hoist room.

The hoist is well guarded and the installation is maintained in a cleanly manner; this hoist is also equipped with 220-volt a.c. control.

[illegible]

The three hoists are equipped with indicators so that the hoistman may know at all times the exact location of the trip.

Underground Substations

There are three underground electrical substations. Two are on the main slope near 9 left and 17 left entries, respectively, and one is on the west slope between 2 left and 3 left entries.

The 9th left substation consists of three motor-generator sets, all of the same type and capacity. The motors are 150-hp. and operate on 2,300-volt alternating current; the generators are 100 kilowatts and produce 250 volts direct current.

The 17 left substation consists of one unit, a 2,300-volt a.c., 300-kilowatt, synchronous motor which operates a 200-kilowatt, 250-volt d.c. generator.

The west slope substation consists of a 2,300-volt a.c., 300-hp. motor and a 250-volt d.c. generator.

The three substations are housed in fireproof rooms equipped with steel doors, held open by small hemp ropes extending over the electrical apparatus; in case of fire, the doors will close automatically by weights and by gravity as soon as the ropes are burned through.

Both rock dust and suitable fire extinguishers are kept at each substation on the intake air side. Rubber mats are provided for switch panels, and all switches and wiring are installed in a safe and efficient manner. These stations are kept exceptionally free from loose flammable material; metal receptacles are provided in each substation for such material.

The one permanent pumping station is near the bottom of the main slope, and there is a temporary installation near the face of the west slope.

The main-slope station is thoroughly fireproofed, and provision is made for the closing of the doors automatically in case of fire.

Telephones are installed at suitable points throughout the mine, and connect with the office and surface plant; the wiring is in good shape and is kept at a safe distance from trolley and other circuits. A code of signals is posted at each telephone.

Signal wires on both the main and west slopes and on the incline carry 15 volts; small transformers in the hoist rooms convert 220 volts a.c. to 15 volts. Simultaneously with the ringing of the bell, when a signal is given, a light flashes in the hoist room.

Every entry is provided with a sectional switch near its junction with the slopes, which permits the isolation of any entry without interfering with other circuits in the mine.

EXPLOSIVES

Hercoal C, a permissible, 60 per cent weight strength and 6.7 per cent cartridge strength (or bulk strength), 1½ by 8 inch explosive, and No. 6 instantaneous detonators are used exclusively for blasting. Shot firers load and tamp all holes. In certain sections the miners drill holes, make up the primers, and place the explosive convenient to the face for the shot firers; in other sections, company men do the drilling. If in the judgment of the shot firers a hole is not properly placed, they refuse to shoot it.

The detonator is inserted into the side of the cartridge, and a half hitch is taken around the cartridge with the detonator legs. All detonator legs are "shorted" at the factory by twisting the bare ends of the wires together.

Wooden tamping bars and adobe stemming are used. Not more than six holes are fired in any working place, and the permissible limit of 1½ pounds of explosive per shot is not exceeded; the writer was advised that it was rarely necessary to equal the permissible limit per charge.

Detonators and explosive are carried underground separately by the shot firers and miners in specially constructed, rubberized canvas bags, equipped with two shoulder straps which permit the bags to rest comfortably on the back. There are no power circuits on the slope, with the exception of the signal wires.

All shots are fired electrically from outside the mine, with no one underground, using 220 volts a.c.

No. 6 wire is used on the blasting circuit, except in rooms, where No. 14 wire is used. There is a switch in the circuit on each entry near the slopes which is kept locked when not in use. In addition, there are three switches in the shot-firing circuit near the manway portal; two of these are double-pole knife switches, and the third is operated by a pipe gate. The gate is locked open until all shot firers are out of the mine; it is then closed and locked. When the two additional switches are closed, the circuit is completed between the mine and the master or time-limit switch in the nearby check cabin. (See fig. 1 for details of time-limit switch.)

Each of the three shot firers has a key to only one of these switches; this makes it necessary that all of the shot firers be out of the mine prior to blasting. In other words, the circuit can not be completed in the mine until the shot firers reach the surface, unlock their respective switches, and close them.

After these three switches have been closed, the time-limit switch is then unlocked and closed momentarily; this switch is of the solenoid type and is made up of three switches. The main switch is of the 2-pole type; prior to closing it, a plunger operating in a slotted 1-inch pipe is raised to the top of the slot and connected to the solenoid switch, which automatically releases the plunger when the double-pole switch is closed. The plunger drops 6 feet 8 inches and opens another switch which breaks the main circuit in the event the double-pole switch is still closed; it is therefore impossible to maintain the current on the firing circuit for a longer period than it takes the plunger to fall. In this case the time is considerably less than one second. (See fig. 1 for details of time-limit switch.)

VENTILATION

The mine is ventilated on a split system by a Jeffrey fan, running at 330 r.p.m., and exhausting about 163,000 cubic feet of air per minute against a water-gage pressure of 3.75 inches.

There are several openings to the surface; the two principal intake airways are the main slope and the manway. Each side of the mine is on a separate return; and all returns join some distance inby the fan. The west slope section is on a separate split; the 9 left and southeast entries and air courses serve as intakes and returns for this section.

Basing conclusions on a number of air readings in both intakes and returns, and on the character of the air analyses, it may be said that this mine is exceptionally well ventilated. The amount of air in circulation through the last open crosscuts varied from 6,000 to 18,200 cubic feet per minute.

Stoppings are substantially constructed of slack coal and cement; after they are thoroughly set they are then gunited, which generally takes care of shrinkage and insures an air-tight joint between the stopping and ribs and roof.

Overcasts are constructed of reinforced concrete, using broken stone or gravel. The cross sections of the overcasts constructed during recent years are exceptionally large, and are comparable to the cross sections of the airways served.

Used sail cloth of heavy canvas serves as curtains on entries to deflect the air into the rooms; similar material is employed in room crosscuts. Line brattice is used to the faces of entries and air courses to keep them free of methane. Well-constructed wooden doors

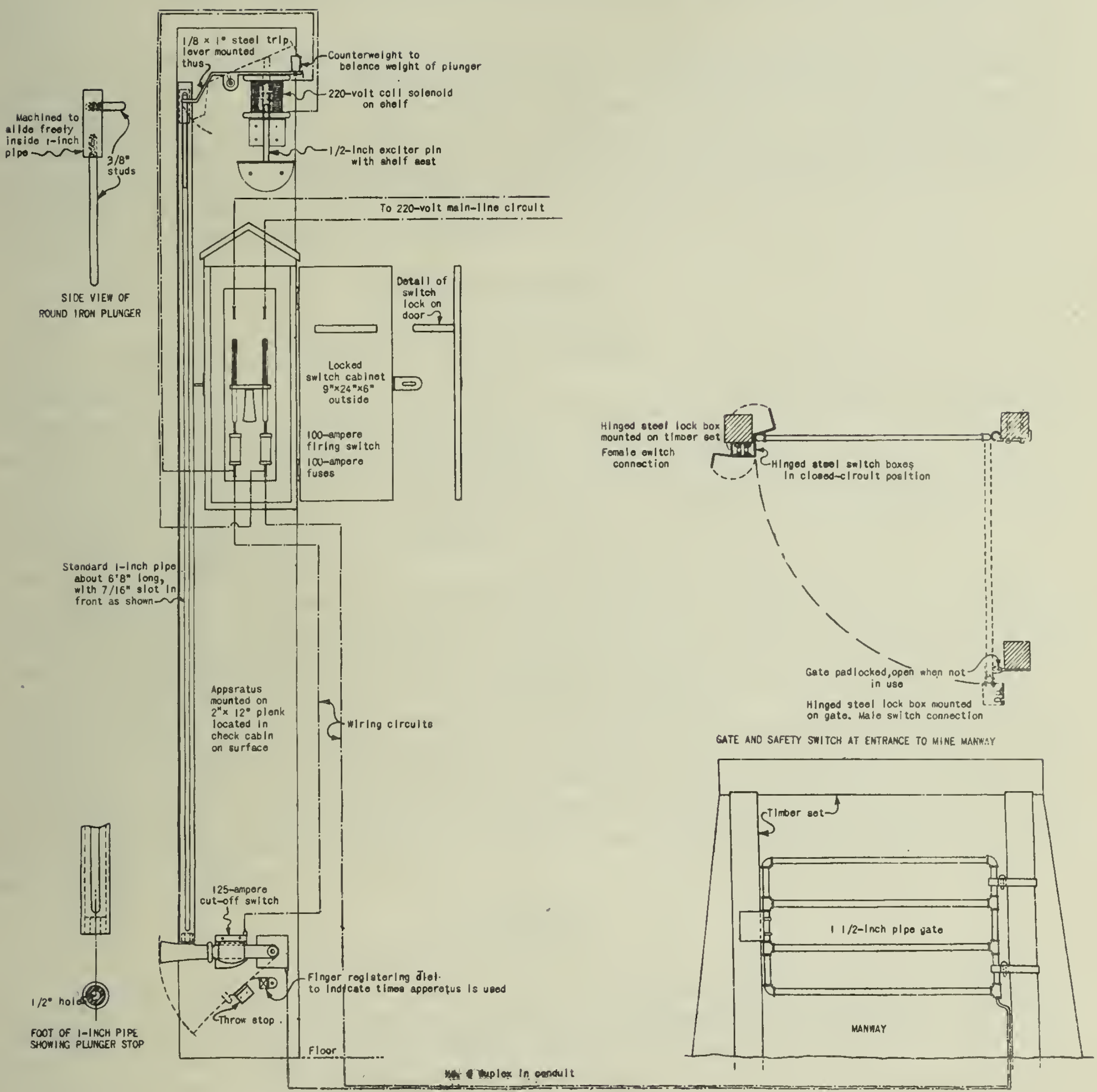

Figure 1.- Time-limit switch layout for shooting system, also gate and safety switch. Note: Time limit is governed by length of fall of plunger; in this case less than 1 second is required

are used to a limited extent, and when on trolley-locomotive haulage ways, care is taken that the doors do not come in contact with trolley wires, which are 6 to 7 feet above the rail for the most part and are suitably guarded with hinged boards and fire hose at points where contact is most likely.

Analyses of air samples recently taken in the mine showed the presence of methane, although in relatively small quantities (0.08 to 0.28 per cent) due to the effectiveness of the ventilation.

Two samples taken in the main return of the mine 40 feet inby the fan contained 0.15 and 0.16 per cent methane in 162,648 cubic feet of air per minute, or 363,000 cubic feet of methane liberated every 24 hours. This volume of methane diluted to its maximum explosive point of approximately 9 per cent, would equal 4,033,588 cubic feet of explosive mixture every 24 hours, a volume of explosive mixture sufficient to fill more than 10 miles of entry, 6 by 12 feet in cross section; this indicates that a dangerous condition may exist if the ventilation should become materially deranged.

MOISTURE CONDITIONS

Moisture conditions throughout the mine are excellent. Due to the pitch of the bed, natural drainage is good.

All loaded trips are sprinkled at the slope partings, and the empty trips are thoroughly wet down near the winding engine upon their return from the tipple.

The rule requiring water on the cutter bars of mining machines is strictly enforced. Coal piles are wet down, and faces are sprinkled prior to blasting. The services of three men are required in connection with sprinkling. All working places are kept wet between the last open crosscut and the face.

Two reservoirs are maintained in the abandoned workings on the left side of the mine, so placed that the danger from an inundation due to possible pillar failure is remote. The water impounded in these reservoirs is available in case of fire and for use in sprinkling and on cutter bars. The pumps discharge into the reservoirs, and when they are full the valves automatically close and the water discharges on the surface.

Tests made by the United States Bureau of Mines show that Utah coals will not absorb enough moisture to prevent the propagation of an explosion, but if kept sufficiently wet, the dust will not get into suspension; this condition will minimize the chances of initiating a coal-dust explosion, or even of extending or propagating one which may have been started by explosive gas or otherwise.

ROCK-DUSTING

An impure limestone from Devils' Slide, Utah, is used as rock dust; this material contains about 75 per cent of limestone and 25 per cent of impurities, such as silicate minerals. Inasmuch as this dust does not readily absorb moisture and therefore has little or no tendency to cake, it appears to be well adapted to mine use.

All open, accessible workings are rock-dusted, and prior to removing the track from abandoned workings such places are again thoroughly rock-dusted up to the face. Each working place, including all rooms, is rock-dusted up to the last crosscut, or to the watered zone.

During 1931, 0.98 pound of rock dust was applied to the mine for every ton of coal produced. This amounted to 306,149 pounds, or 153 tons of rock-dust.

The rock dust is applied by both high and low pressure machines; a cement gun is used to apply the dust under high pressure and is operated by a portable, electrically driven compressor. The low-pressure machine was made in the company's shops, and its operation has been entirely satisfactory.

Tests made by the United States Bureau of Mines show that there should be at least 70 per cent of incombustible material (moisture plus ash) where no methane is present, and 75 per cent inert material where 1 per cent of methane is present, to prevent the propagation of an explosion in Utah coals; results of Bureau of Mines experiments on the explosibility of Utah coal dust may be found in Technical Paper 386, "Explosibility of Coal Dust from Four Mines in Utah," published in 1927.

Rock-dust barriers are installed in both entries and air courses near the slopes from which they are turned; and in addition, barriers are installed in the connections between entries. The life of the barrier usually equals that of the entry. All barriers at this mine were found in good repair, and the dust was dry.

ILLUMINATION

The slopes, partings, hoist rooms, pump stations, and substations are lighted with incandescent bulbs using 250 volts direct current.

All underground employees use permissible cap lamps; and flame safety lamps, approved type, are used for testing purposes. Each machine crew is provided with a flame safety lamp, and the rule requiring the machinemen to test for gas prior to taking the machine into the face is rigidly enforced. The machinemen are required to have credentials equivalent to those of the fire bosses. In exceptionally gassy places, tests are made for gas during the process of cutting.

ACCIDENT PREVENTION

Although the company maintains no specific safety organization, safety is considered a major operating problem. The underground officials, through constant supervision and personal contact with the workers, are endeavoring to make the entire organization safety-minded.

The company is a member of the Spring Canyon Mine-Rescue Association. In case of emergency, there are available 40 trained apparatus men, comprising eight teams, in the employ of this company. Four of the eight teams have had actual experience is rescue and recovery operations at time of fire or explosion.

During April, 1932, the United States Bureau of Mines accident-prevention course was given to 137 employees of the company; the work of training all of the employees in first aid was completed in May, 1932. First-aid material is kept at suitable places underground.

A mass safety meeting was held at the completion of the first-aid training, and the management plans to hold such safety meetings each month in the future.

An up-to-date hospital is maintained at Spring Canyon, with a surgeon and trained nurse in constant attendance.

The accident record for 1931 may be briefly stated as follows:

Total accidents		168
Total coal mined	tons	312,397
Coal mined per accident	do.	1,859
Time worked	man-hours	516,976
Total time lost (one fatal = 6,000 days)	days	8,387
Total time lost per accident	do.	49

$$\frac{168 \text{ accidents} \times 1{,}000{,}000}{516{,}976 \text{ man-hours}} = 324, \text{ or frequency rate.}$$

$$\frac{8{,}387 \text{ days lost} \times 1{,}000}{516{,}976 \text{ man-hours}} = 16.4, \text{ or severity rate.}$$

During the 12-year period 1918-1929, the average number of fatal accidents per year was 2.91, as against 1 in 1931; the tonnage produced per fatality from 1918 to 1929, inclusive, was 139,830, compared with 312,397 tons produced to 1 fatality in 1931. The fatality rate of this company per 1,000 employees during the period 1918-1929, inclusive, was 7.8, which was materially lower than the 9.4 rate for all of Utah's coal mines for that period but was very much higher than the similar rate of 2.9 for the coal mines of the United States for the same period.

While there is considerable room for improvement in accident rates, a material reduction can confidently be expected if the increased supervision and the educational work which has been instituted is forwarded. Strenuous effort is being made to create and maintain a vital interest in safety on the part of the individual employee, as such interest and enthusiasm are a prerequisite to the success of any safety program; and with the earnest, alert attitude of the management now in effect toward maximum mine safety, there is the best of reasons for the belief that in the near future good results in safety performance will be achieved by this company.

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DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
SCOTT TURNER, DIRECTOR

INFORMATION CIRCULAR

METHOD AND COST OF MINING SAND AND GRAVEL AT THE FARMINGTON (CONN.) PLANT OF THE ATLAS SAND, GRAVEL AND STONE CO.

BY

JOHN S. DUNNING

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

METHOD AND COST OF MINING SAND AND GRAVEL AT THE FARMINGTON (CONN.) PLANT OF THE ATLAS SAND, GRAVEL AND STONE CO.[1]

By John S. Dunning[2]

INTRODUCTION

This paper is one of a series describing methods and costs of recovering sand and gravel from deposits in the United States and deals directly with the operations of the Atlas Sand, Gravel and Stone Co. at Farmington, Conn.

These papers are designed to disseminate technical information regarding the methods used. The cost tabulations represent local expenditures only and not total production costs. It is recognized that publication of total costs might in some instances cause embarassment to individual producers, as well as to the industry as a whole. On the other hand, operating costs are essential to the technical discussion and study of methods employed. The attention of the reader is specifically called to this differentiation in order that no misunderstanding of the scope of the cost tabulations may ensue.

The plant described is a dry-land operation of especial interest because its present methods of handling the material were developed by a series of radical changes each of which brought about greater economy and increased the quality and quantity of the product. The plant can now produce the finished material at a marked reduction from the cost of production in the plant as it was first built.

ACKNOWLEDGMENTS

The author wishes to acknowledge the assistance of S. N. Dunning, president of the company, in furnishing data, and of J. R. Thoenen, mining engineer of the U. S. Bureau of Mines, who helped in compiling this report.

HISTORY

The Atlas Sand, Gravel and Stone Co. began operations in 1913 at Farmington, Conn. The original plant consisted of three bins, above which were mounted a small jaw crusher and screening and washing apparatus. Material was drawn to the plant by a 1½-yard slack-line cable excavator.

It was not long before costs began to rise on account of the necessity of going further and further away for sand and gravel which was not too deep under water. It was found that the presence of occasional large bowlders on which the bucket would catch limited the use of the cable excavating system to dry material or at best to that in shallow water where the bowlders could either be blasted or removed by hand. The approximate limiting depth of water

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1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6676."

2 - One of the consulting engineers, U. S. Bureau of Mines; and engineer, Atlas Sand, Gravel and Stone Co.

for economical operation of the slack-line cableway was found to be about 5 feet. However, if no large bowlders were encountered, dragging at each position was sometimes continued to a depth of 8 or 10 feet. The limiting depth, in other words, was determined by large bowlders rather than by the water itself. About 1916 the cable system was discarded in favor of a power shovel which loaded trucks. These trucks dumped into a hopper, and an inclined bucket elevator raised the material to the crusher.

About 1917 an effort was again made to reclaim the under-water deposit near the plant by the use of a crane mounted for railroad traction with clamshell attachment.

A line of track was run parallel to one side of the deposit to be worked and the clamshell excavated on that side of the track. Since the deposit is so tightly packed, excavation could be carried quite close to the track. Then the track would be moved back from the excavation and the clamshell would take another strip. This method of operation was not developed because the deposit was found to be too tightly packed to be dug efficiently by the clamshell bucket. Consequently it was soon replaced by the present set-up, consisting of a power shovel and industrial railway system.

During this time the screening tower was rebuilt three times, and enlarged each time, until now it has bin storage space for 8 different sizes of material totaling about 400 tons. All crushers were removed from the tower and placed on the ground, and a belt conveyor is used to elevate the crushed material.

Considerable difficulty was experienced in getting enough water for washing the increasing output of the plant. After several unsuccessful attempts to drill wells to provide sufficient water, an 8-inch pipe line was run about 1,000 feet to the Farmington River, which gave ample washing water for a clean product. This water had to be raised a vertical distance of 22 feet, of which 8 feet was suction head and 14 feet discharge head.

GEOLOGY

Geologically the deposit falls within the area defined by Flint[3] as of glaciolacustrine origin. The materials were deposited on the beds of ancient glacial lakes formed by the impounding of glacial waters behind ice gorges. The many lacustrine terraces occurring in this section are differentiated by their elevation above sea level.

There is a certain amount of horizontal stratification present, although the distribution of sand and gravel is rather uniform throughout the deposit. There are minor variations, however, as exemplified by the occurrence of thin layers of sand with no gravel.

Scattered throughout the deposit and in some instances collected in local clusters are bowlders of soft sandstone. These have evidently been brought to the lake by ice or glaciofluvial waters and left in place as the boundaries of the lakes contracted.

The deposit extends several miles north and south of Farmington, roughly paralleling the Connecticut River lying 10 miles to the east. The tracks of the New York, New Haven, and Hartford Railroad form the west boundary of the property and there is a siding about ½ mile north of the plant. It is one of many deposits of sand with more or less gravel in the northwestern section of the State. The deposits are usually easily recognized by the sparse clumps of sedge and gray birches which with a few cottonwoods and scrub oaks almost invariably form the only vegetation. Figure 1 is a sketch of the land owned by the company and shows the general location of the plant and pit.

The overburden on this deposit is a light sandy loam varying from 1 to 3 feet in depth. Gravel and sand are stratified horizontally, but the bank runs quite uniformly about 40 per cent sand. Occasional bowlders of soft red sandstone occur which would crush into poor ag-

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3 - Flint, R. F., The Glacial History of Connecticut: Bull. 47, Conn. Geol. and Nat. Hist. Surv., 1930, pp. 294.

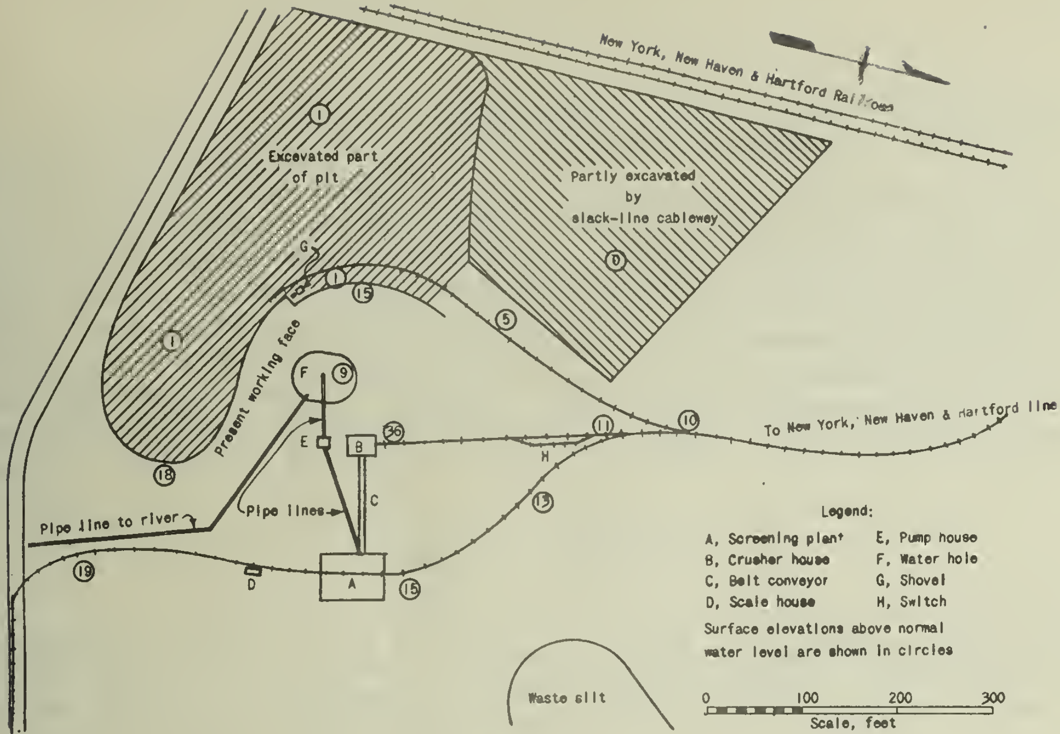

Figure 1.- Map of property

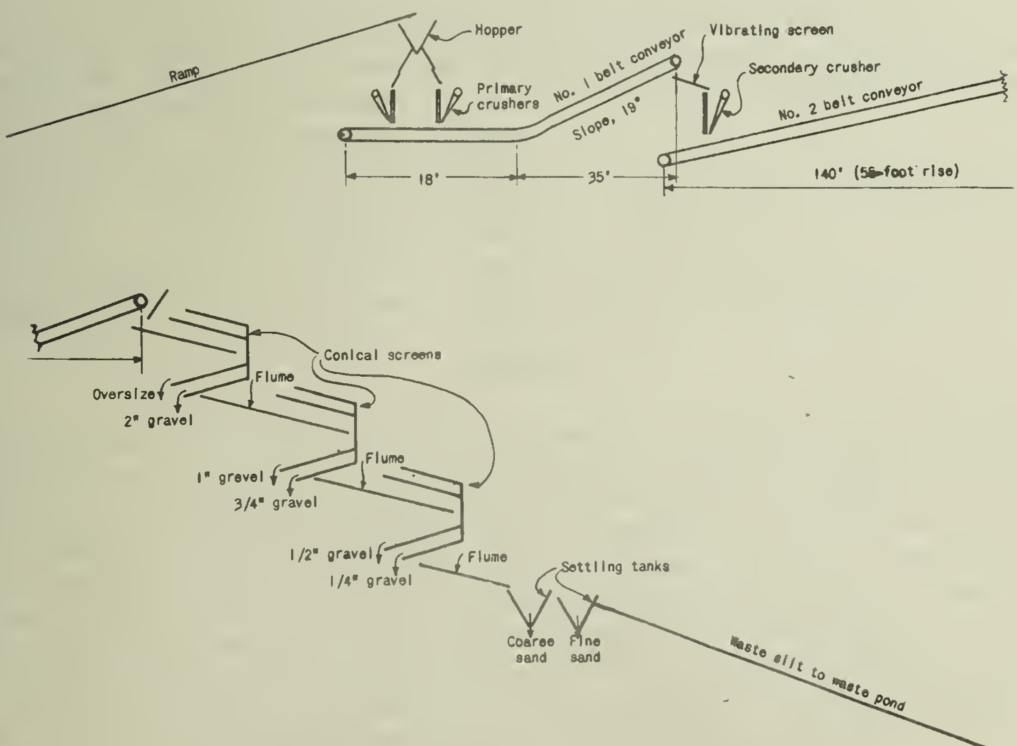

Figure 2.- Flow diagram of plant

regate. Furthermore, these pieces of sandstone are liable to block the crusher since they will not fracture as a brittle stone would, but give without breaking. Consequently they are cast aside at the pit by the shovel.

An abandoned canal crosses the property. This canal is filled with black muck and consequently has been avoided as much as possible. The land rises to the north, and about ½ mile away where it is 45 feet higher the sand is filled with pockets of large pieces of red sandstone. Lying ½ mile further north and another 50 feet higher is a good deposit of clean sand with only 5 to 10 per cent gravel. These three types of deposit are evidently from different terraces, and presumably, of somewhat different ages. All are stratified horizontally.

The sand and gravel bank extends generally 15 to 20 feet above water level and an undetermined depth below water level—at least 30 feet, however — and is uniformly commercial to this depth.

METHODS OF PROSPECTING

Originally a roadside borrow pit disclosed the type of deposit in Farmington. A few hand-dug holes bore out the indications of this pit, and the distinctive vegetation made extensive prospecting seem unnecessary. To date, no part of the deposit has been struck which was not worth processing, except the area where the old canal is and higher up where the sand is filled with sandstone bowlders.

No prospecting costs were kept.

CHOICE OF METHOD OF MINING

Obviously, at first, under-water mining was impossible; so a drag line was used with the intention of dredging as soon as a pond of sufficient depth had been excavated. Excavation by cableway excavator disclosed the fact that the movement of subsurface water to the excavated pit was not enough to supply the plant with wash water and still maintain sufficient depth in which to float dredging equipment.

It was found that by pumping, the water could be removed from the pit, and that when pumping ceased, the water slowly returned to its original level. This lack of sufficient water prevented the use of a dredge, and the cableway excavator could, of course, cover only a limited area; so a power shovel with industrial railway was decided upon as the most economical method of mining the many years supply of material above water level.

STRIPPING

The overburden varies from 1 to 3 feet in depth. It consists of sand penetrated with a diminishing percentage of loam to a depth of about 2 or 3 feet and topped with about 6 inches of turf. It has been proved possible to wash out nearly all the loam in the plant, but to assure a clean product about 1 to 2 feet is usually stripped. The overburden is removed by either the main-pit shovel or a smaller shovel during the slack season. The material is cast into windrows about 15 feet high and of a length varying with the area stripped. The overburden, which has no commercial value, is loaded into trucks when they can be spared from the delivery department, and transported roughly 1,000 feet to low land on the adjoining property. The overburden is always moved from the company's deposit, otherwise it might have to be moved again later. It is also never dumped in the excavation since mining of that part of the deposit which is under water is contemplated in the near future.

EXCAVATION

Excavating is handled with a $1\frac{1}{4}$-yard steam shovel with caterpillar traction. The bank is excavated down to the level at which the water becomes troublesome. This generally results in a face of some 12 to 15 feet. The shovel works along the face, cutting a curved swath about 25 feet wide, and is followed by haulage tracks. When the end of the face is reached, the shovel starts back, the track being shoved close enough for loading.

A medium-grade bituminous coal has been found to be most economical for use in this shovel and in the locomotive to be described in the following section. Coal is delivered to the shovel by a horse-drawn dump cart. Water is drawn from the water hole (fig. 1) and pumped to the shovel through a 1-inch pipe line by a 2-inch centrifugal pump powered by a direct-connected, 5-hp. motor.

The excavating crew consists only of the shovel operator and fireman.

TRANSPORTATION

Two 12-cubic yard steel-lined Western bottom-dump cars transport the raw material from the bank to the hopper above the primary crushers. An 18-ton steam locomotive draws them one at a time to the switch (fig. 1), an average distance of 800 feet. This locomotive also hauls loaded cars to the railroad.

At the bottom of the incline a 1-inch cable is attached to the loaded car and a 2-drum hoist powered with a 75-hp. motor draws it up the single-track gravel incline and over a timbered trestle, a distance of approximately 200 feet. In this distance it rises 25 feet to a 15-cubic yard hopper, where the car is dumped. Dumping is controlled by a handwheel worm and gear which slackens the chains holding the bottom doors. After the doors are wound up again, the car is allowed to coast down the incline, braked by the hoist, to the switch, where it is picked up by the locomotive and the cable transferred to the other car.

The track used is 55-pound standard gage, and special effort is made to get chestnut ties, since chestnut will last through the continual wetting many times better than other local woods.

CRUSHING

Stones larger than 10 inches in diameter are caught on a bar grizzly. This grizzly, 8 feet wide by 15 feet long, consists of standard 55-pound rails, with the ball uppermost and set on the same slope as the trestle. The large stones are removed by hand. The hopper below is constructed of 3 by 6 inch oak planks bolted to braces and is lined with steel sheets. The sides slope at an angle of 55°.

Two chutes, each controlled by a 24 by 30 inch slide gate, lead the sand and gravel to two 14 by 26 inch jaw crushers, arranged tandem fashion over a conveyor belt. Each crusher is preceded by a bar grizzly constructed of narrow-gage car rails, with 2-inch spacing through which the sand and fine gravel drop directly onto the belt, thus providing a cushion for the discharge of the crusher and eliminating unnecessary wear on the belt.

The two primary crushers are run with a maximum opening of about 4 inches. They discharge directly onto No. 1 belt conveyor. Figure 2 is a flow sheet of the plant.

A mechanical vibrating screen takes the discharge from No. 1 belt conveyor and separates the plus $2\frac{1}{4}$-inch stone to go through the secondary crusher while the fines drop directly to No. 2 belt conveyor. This vibrating screen is one of the company's own make, using roller bearings and having a $\frac{1}{4}$-inch throw. It is driven by an 1,800-r.p.m., 5-hp., ball-bearing motor through a single V-belt drive which cuts the speed of the screen shaft to 800 r.p.m.

This shaft is 3-7/16 inches in diameter and is supported in bearings on each end. Equidistant on the shaft are two eccentric bearings connected directly to the screen. The eccentricity is ¼ inch in a vertical circle. The screen cloth is made of ½-inch round wire with 2-inch square openings and has an average life of 100,000 tons.

The secondary crusher is a 36 by 6 inch jaw crusher, which is closed to a 2¼-inch or smaller opening. The crusher straddles No. 2 belt conveyor and discharges directly onto it.

Any necessary recrushing is handled by loading the oversize into a car under the bins, from where the locomotive takes it to the hopper above the primary crushers, to be sent through the plant again. Often the 2-inch and 1-inch sizes are recrushed in this manner to make more of the smaller sizes.

CONVEYOR SYSTEM

As shown in Figure 2, No. 1 belt conveyor runs under the two primary crushers horizontally for a distance of 18 feet, then rises on a slope of 19° for a distance of 35 feet to the vibrating screen. This slope is about as steep as is practical with this gravel. It is driven by a 15-hp., ball-bearing motor through a 6-inch, 4-ply, flat rubber belt.

No. 2 conveyor is 140 feet between centers and rises 55 feet, with 36-inch diameter head and tail pulleys. A 25-hp., ball-bearing motor drives No. 2 belt through an 8-inch, 6-ply, flat rubber belt.

Both conveyors use 36-inch, 6-ply, rubber-covered canvas belting. The rubber covering is 1/8-inch thick on the wearing side and 1/16-inch thick on the bottom. Both conveyors run at a speed of 175 feet per minute. No. 1 conveyor belt has a life of approximately 150,000 tons and No. 2 conveyor belt of approximately 500,000 tons.

WASHING AND SCREENING PLANT

At the discharge end of No. 2 belt a 4-inch stream of water is shot into the sand and gravel and aids its flow down two chutes to two parallel screen banks. More water is added in spray pipes in the screens. The total water used is about 500 gallons per minute. Each bank consists of three double conical screens, arranged to pass six sizes of material, the fines of each screen being chuted to the next screen. These sizes are as follows:

Passing	Retained on	Commercial rating of gravel
2½-inch round	1¾-inch round	2-inch
1¾ do.	1¼ do.	1-inch
1¼ do.	7/8 do.	¾-inch
7/8 do.	3/8 do.	½-inch
3/8 do.	¼ by ½ inch slots	¼-inch
¼ by ½ inch slots	-	Sand

The conical screen has several unique features. As shown in Figure 2, the gravel is introduced at one end of the screen and travels back toward the other end over the screen. The main advantage of this type of construction is that the larger sizes are disposed of first. Since the larger sizes compose a high percentage of the total and also since they require less washing than the small sizes do, it is ideal to be able to eliminate them on the first few screens. A single cylindrical screen would carry the large sizes through the whole length of the screen.

There are disadvantages to the conical screen, chief of which is the headroom required, since the second screen must be set below the bottom of the first.

The screen head carries all the weight of the screen and the gravel in it, transmitted through angle-iron braces from the screen head. The head is a casting of the company's own design mounted on a 4-inch shaft 10 feet long. Four 6 by 6 inch angle-iron braces serve to keep the screen plate rigid. The inside screen plate is braced to these angle irons by U-shaped pieces, keeping the spacing between the two screens at 6 inches.

A slope of only 6 inches in 6 feet is used in order that the material will travel out of the screen slowly enough to insure thorough scrubbing and good separation.

The life of the screens varies considerably, but to avoid repairs in mid-season, they are generally changed at the end of each operating season, except when operation is very irregular, as at present. Probably the average life of the screens is about 150,000 tons.

WASHING AND SCREENING

Water is sprayed into each set of screens at a pressure of about 40 pounds by a 2-inch pipe with 1/16-inch perforations. The whole screen revolves in a sort of bathtub lined with sheet zinc which collects the passed fines and water to be chuted to the next screen. Zinc is used because it is less liable to corrosion, and its use obviates frequent replacements and high labor costs.

All screens in both banks are run at 20 r.p.m. by a 15-hp. motor through speed-reducing gears, sprockets, and roller chains.

The sized gravel is discharged either directly or through a short chute to the proper bin. There are no facilities for mixing the different sizes. Slide gates, 12 by 18 inches in size, operated by side levers from the ground, control the discharge of the washed and sized material to either trolley freight, trucks, or railroad cars. The sliding, on the New Haven, Northampton branch of the New York, New Haven and Hartford Railroad is about half a mile north of the plant.

SAND-SETTLING TANKS

Sand passing through the $\frac{1}{4}$ by $\frac{1}{2}$ inch mesh of the last screen is flumed over two settling tanks. While passing over the first tank the larger and heavier grains of sand settle out. This is all minus 3/8-inch and runs from 0 to 5 per cent minus 50 mesh. Going into the second tank the stream slows down past a series of three baffles which cause the finer sand particles to drop out, leaving only silt suspended in the water which is flumed to the waste field, as shown in Figure 2. The fine sand is 100 per cent minus 10 mesh and from 50 to 60 per cent minus 50 mesh.

These settling tanks are of our own make and design and are shown in Figure 3. They both discharge sand directly to bins below.

WATER SUPPLY

Water requirements for washing and screening are about 500 gallons per minute. In the spring and fall, a water hole by the plant will supply this, but during the summer water is pumped from the Farmington River into the water hole. An 8-inch centrifugal pump driven by a direct-connected, 40-hp. motor is located at the river bank and pumps through an 8-inch pipe line a distance of about 1,000 feet, discharging into the water hole. Another set-up of a similar pump and motor raises the water 57 feet to the screening tower through an 8-inch pipe line paralleling the belt conveyor. At the top of the belt this pipe is tapped and reduced as previously described.

SAND AND GRAVEL STORAGE

A hole 12 by 12 inches has been cut in the side of the coarse-sand bin near the top and a flume has been connected at that point. Whenever this bin gets full, the wet sand washes down this flume to the coarse-sand storage pile.

There is no storage of fine sand at this plant, as only a little of it is produced here.

Gravel in excess of immediate requirements is drawn from the bins by truck and dumped in piles nearby. It is restacked with a ½-yard gasoline shovel into higher piles so as not to cover too much ground. Both the sand and gravel are loaded out of storage with the same shovel.

PERCENTAGES OF VARIOUS SIZES PRODUCED

Quantities of the different sizes produced vary considerably, chiefly due to variation in the deposit and to changes in settings of the crushers to meet market requirements. Also, occasionally, the sizes of holes in one or two of the screens have been altered somewhat in an effort to produce sizes in demand. Again the 3-inch and 2-inch sizes of stone when not in demand are often recrushed. Roughly, however, the percentages retained on screens with openings shown are as follows:

Retained on	Commercial size	Percentage of total
2½-inch	3-inch gravel	3
1¾-inch	2 do.	7
1¼-inch	1 do.	20
7/8-inch	¾ do.	16
3/8-inch	⅜ do.	8
¼ by ½ inch slots	¼ do.	3
Classifier	"B" sand (coarse)	30
Settling tank	"A" sand (fine)	10
Silt	-	3
		100

EMPLOYEE PAY SYSTEM

All employees are paid by the hour except the foreman and superintendent, who draw weekly salaries. Figure 4 is a graphic chart of the organization.

SAFETY METHODS

Safety work is carried on through the foreman, who is continually watching to avoid dangerous situations. A good safety record has been established by the company.

First-aid supplies are always on hand and use of antiseptics for any skin abrasion is compulsory.

Table 1.– Detailed Costs

Name of pit: Atlas Sand, Gravel and Stone Co., Plant 1.
Period: Jan. 1, 1931, to Dec. 31, 1931.
Total sand and gravel mined during period: 49,840 tons.

Operating costs per dry ton of sand and gravel mined

Operation	Labor	Supervision	Power[1]	Fuel	Repairs[2] supplies	Total
Stripping (loading)	$0.0018	$0.0002	–	$0.0004	$0.0005	$0.0029
Stripping (transportation)	.0026	.0001	–	.0007	.0012	.0046
Loading (gravel)	.0174	.0010	–	.0049	.0069	.0302
Transportation (gravel)	.0321	.0020	$0.0035	.0032	.0062	.0470
Crushing, primary	.0105	.0017	.0130	–	.0066	.0318
Crushing, secondary	.0021	.0010	.0087	–	.0049	.0167
Conveying	.0021	.0005	.0056	–	.0048	.0130
Washing and screening (including water supply)	.0055	.0020	.0152	–	.0107	.0334
Storage bins and stock piling	.0114	.0010	–	.0031	.0052	.0207
Miscellaneous	.0064	.0010	.0015	–	.0050	.0139
Totals	0.0919	0.0105	0.0475	0.0123	0.0520	0.2142

1. Electric power is purchased on a sliding scale, which during this year resulted in an average price of $0.027 per kilowatt hour.
2. Includes repair labor.

Table 2.- Summary of Costs in Units of Labor, Power, and Supplies

Name of Pit: Atlas Sand, Gravel and Stone Co., Plant 1.
Period covered: Jan. 1, 1931, to Dec. 31, 1931.
Material loaded during period: 49,840 tons.

Overburden: 3,000 cubic yards. Weight per cubic yard loose: 2,300 pounds.
Sand and gravel: 49,840 tons. Weight per cubic yard loose: 3,100 pounds.

	Stripping	Mining	Crushing	Other	Total
A. Labor: (Man-hours per ton, sand and gravel)					
Loading	0.004	0.038	0.035	0.047	0.124
Transportation	.007	.098	–	–	.105
Miscellaneous	.001	.010	–	–	.011
Supervision	.001	.004	.005	.0045	.014
Total labor	.013	.150	.040	.051	.254
Average tons per man per shift	–	–	–	–	39.3
B. Power:					
Total power:					
1. Shovels ... hp.hrs. per ton	.008	.620	–	–	–
2. Locomotives ... do.	–	.248	–	–	–
3. Pumping water ... kw.h. per ton	–	–	–	.371	–
4. Car hoist ... do.	–	–	–	.129	–
5. Crushing ... do.	–	–	.804	–	–
6. Conveying ... do.	–	–	–	.208	–
7. Screening and washing ... do.	–	–	–	.192	–
8. Storage ... hp.hrs. per ton	–	–	–	.122	–
9. Shop and lighting ... kw.h. per ton	–	–	–	.055	–

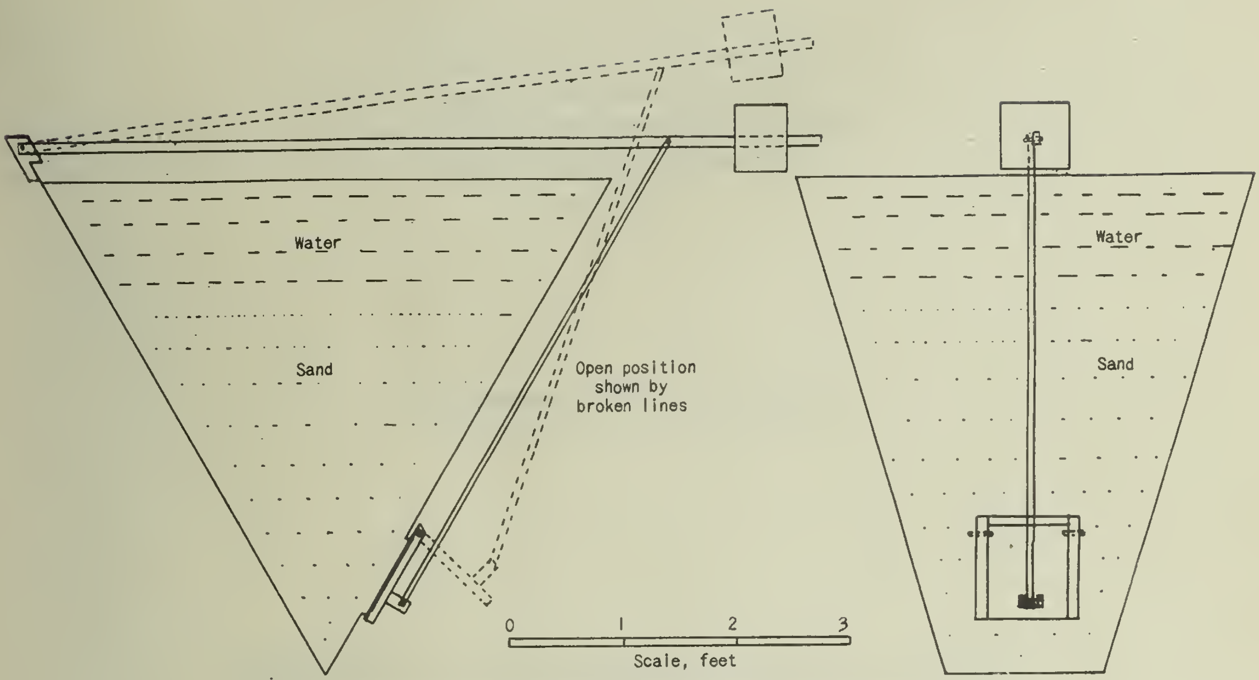

Figure 3.- Sand settling tank

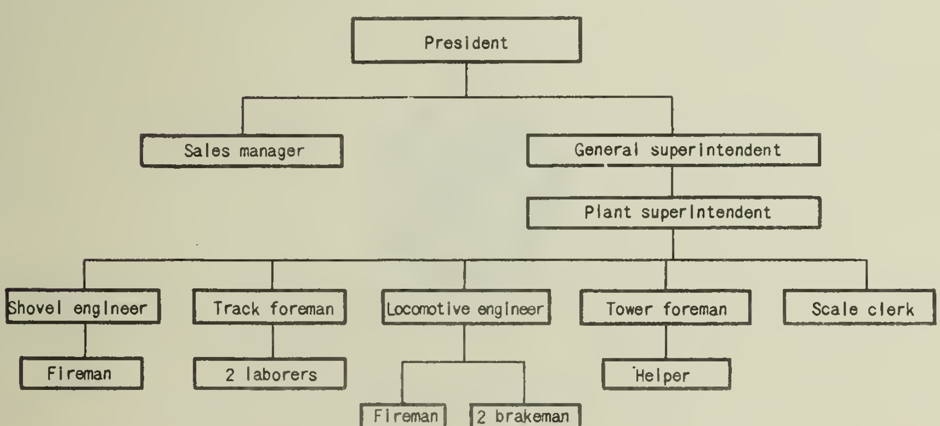

Figure 4.- Organization chart

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January, 1933

Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

WORKING AN UNDERGROUND MINE SIX YEARS WITHOUT LOST-TIME ACCIDENTS

BY

C. A. Herbert

I.C.6677,
January, 1933.

INFORMATION CIRCULAR

DEPARTMENT OF COMMERCE - BUREAU OF MINES

WORKING AN UNDERGROUND MINE SIX YEARS WITHOUT LOST-TIME ACCIDENTS[1]

By C. A. Herbert[2]

The mine of the Alpha Portland Cement Co., Ironton, Ohio, working in a bed of limestone 96 feet thick at a depth of approximately 510 feet below the surface, has established a wonderful safety record, having operated 6 years--from September 21, 1926, to September 21, 1932 - without a lost-time accident. This mine, with 61 to 86 employees (average 68) and daily output of 1,000 to 1,200 tons of limestone, operated 1,041 days during the 6-year period, or the equivalent of 578,894 man-hours of exposure.

Immediately above the limestone is a 20-foot bed of hard sandstone. At present about 10 feet of limestone is left up as a roof and about 42 feet is being extracted. Of the lower portion of the bed it is believed that about 25 feet immediately below the present workings is suitable for cement and will be extracted later.

The mine workings are reached by two rectangular concrete shafts, one a 2-compartment hoisting shaft and the other a 2-compartment air and escape shaft. In the latter shaft one compartment is used as an airway and the other as a cage compartment for a man and material cage.

The tipples at both the air shaft and hoisting shaft are of steel. The ground landings at both shafts are fenced in with hand-operated gates in front of the cage compartments.

The two shafts and the crushing plant, which is a part of the hoisting shaft tipple structure, are adjacent to the mill buildings.

The rock is hoisted on self-dumping cages in 4 ton capacity, end-gate, steel cars.

An electric hoist with automatic overwind device is used at both the main shaft and air shaft. The controls for the main hoist are in the tipple where the engineman is stationed and from which point he has an unobstructed view of both the ground landing and the cage coming into the dump.

PRODUCTION AND UNDERGROUND EMPLOYMENT

The mine produces 1,000 to 1,200 tons per day of 8 hours and the plant is working at full capacity. There are now 61 employees, but as said, the average employment was 68 during the 6-year period.

LIGHTING

The men use carbide lamps. The mine bottom, switches, and partings are lighted by electricity, and the electric shovels have flood lights.

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from the U. S. Bureau of Mines Information Circular 6677."

2 - District engineer, U. S. Bureau of Mines Safety Station, Vincennes, Ind.

VENTILATION AND METHOD OF MINING

The mine is developed by the room-and-pillar method, and unlike many other limestone mines it has a well-planned ventilation system. The rooms are turned from entries in which the crosscuts have been closed with concrete brattices to create a definite circulation of air; these are also built in rooms where necessary to force the air along the faces. The mine is dry and of even temperature throughout the year, and the working conditions are excellent.

Approximately 75,000 cubic feet of air per minute is forced down the air shaft by a motor-driven centrifugal fan on the surface. A duplicate fan is installed at the bottom of the air shaft as a reserve.

As originally opened, rooms and entries were about 27 feet high and about 25 feet wide, but as the stratum being left for a roof did not stand well, the height of rooms was increased to about 41 feet to reach a strong, hard stratum of rock which gives little or no trouble from roof falls; the width of the working places has also been increased to 35 feet. The company is now starting back through the old working places and enlarging them to the dimensions given; this is accomplished by first shooting off the ribs, which fills up the place sufficiently with broken rock to enable the men to work on top of it to drill and shoot the top.

In the advance work, mining is by the heading-and-bench method, using two benches. The heading is driven 16 feet high, the top bench 16 feet high, and the lower bench 9 feet high. The heading and top bench are usually advanced two rounds or 18 feet before the bottom bench is lifted by 18-foot horizontal shots.

The rock is loaded into the cars by means of Thew electric tractor-mounted shovels operated on 250 volts d.c.

Drilling is done with tripod-mounted hammer drills operated by compressed air.

BLASTING

All blasting is done with 40 per cent ammonia dynamite, no-delay and first and second delay detonators and blasting battery, by a crew which starts charging holes at noon; no shooting is done until the day shift is out of the mine. The shot holes are stemmed with sand-filled dummies. As all shots in any one place are not fired simultaneously, it is necessary for the blasting crew to return to the face during blasting; however, the places are shot in rotation and enough time elapses between the firing of the different rounds in any one place to permit the smoke to clear up.

Only one day's supply of explosives is allowed in the mine; the explosives are taken into the mine in a wooden car hauled by mule and are kept in a locked magazine until needed by the blasting crew. Detonators are taken into the mine separately from the explosives and kept in a special magazine.

The rock is well stratified and breaks up well when shot; it is seldom necessary to do secondary blasting; but when it is, one of the crew that scales off the loose rock from roof and ribs is assigned to this work. Secondary shooting is done during the shift by drilling holes in the rock and using light charges of dynamite and fuse.

Misfires are infrequent; when one does occur, the stemming is withdrawn with a copper needle, a new primer is placed, and the hole is again shot. In loading the holes originally, the primer is put at the bottom of the hole so that there is essentially no danger of striking the detonator when withdrawing the stemming. There has never been an explosives accident at this mine.

HAULAGE

Mules and one storage-battery locomotive are used in gathering at the face; haulage is done by storage-battery locomotives. The track is exceptionally well laid, with 35-pound steel on heavy wood ties to a 38-inch gage. All motor trips are handled at low speed, and as no tripriders are worked it is necessary for the motorman to stop the trips to throw switches. At the junction point of the main and side entries a trapper is employed to control the ventilation doors and to act as a trip despatcher.

The absence of loose rock along the floor of the rooms and haulageways was particularly noticeable, and except at the face where the rock was being loaded, the mine workings were generally clean from wall to wall.

CONTROL OF HAZARDS FROM FALLS OF ROCK

Because of the height of the workings, probably the greatest hazard is from falls of rock from roof or ribs. The roof is extremely good after the loose rock caused by blasting has been scaled off; with the roof about 41 feet above the floor, this matter of removing loose roof material naturally offers a considerable problem.

Eight men are employed to keep the roof and ribs safe from loose rock. Iron ladders are used for inspection and scaling of the ribs up to about 29 feet; to inspect and scale the roof and the ribs above 29 feet, two electrically operated derricks or cranes mounted on tractors are used, the power being supplied through trailing cables.

The boom of the crane is built like an extension ladder and can be lengthened or shortened as desired. It can also be swung from side to side or raised or lowered by power controls. On the outer end of the boom there is a platform about 4 feet square with an iron pipe railing extending around it on which the men work while trimming off the loose rock.

The use of these long ladders for trimming the ribs or walls appears to be extremely hazardous; it is remarkable that men can swing a sledge hammer or handle a bar while perched on top of one of these ladders, without sooner or later meeting with disaster, but apparently they become expert and are able to protect themselves from accident. The blasters must also use ladders in getting up on the benches, which adds to the hazards of the mine; but here again the practice appears to be well safeguarded as to actual occurrence of accidents.

After the face has been blasted, a careful inspection of the roof and ribs is made and any loose rock is pulled down before anyone is allowed in the place.

Five or six feet down from the roof a stratum of dark limestone, known as "black stone," occurs interspersed with vertical joints that not only cause it to be affected by blasting but also to weather badly along the haulage and traveling roads where it is exposed to the air. As a result, this stratum of rock was continually spalling and created a severe hazard due to its height above the floor. To overcome this danger, the company decided to gunite this band of rock along the ribs or walls in both rooms and entries. The guniting proved so effective that it is now applied to the freshly exposed band of rock as the faces of rooms and entries advance, and has apparently removed the hazard completely.

Twice each year the ribs and roof of all the working sections of the mine are given a careful inspection.

ELECTRICITY

The underground electrical equipment is operated on 250 volts d.c. taken into the mine in armored cables at the air shaft. All power lines underground are carried along the roof with drops at convenient points for attachment of trailing cables. Heavy rubber-covered

trailing cables are used and are maintained in excellent condition.

SAFETY ORGANIZATION

The safety program of the mine is a part of the general plant program and is comparatively simple.

Each month a safety meeting of all supervisory officials is held under the leadership of a permanent chairman--the mine superintendent. At these meetings a temporary chairman is chosen to preside at the following meeting, and he, with another official who is chosen at the same time, makes a thorough inspection of the entire plant in the intervening period and reports any dangerous conditions or practices that may have been observed; at the next meeting this report is discussed, as well as information on accidents that may have come in from the Portland Cement Association, the National Safety Council, or other sources.

No special safety engineer is employed at the mine, neither are any general safety meetings held, nor is there an employee safety organization. The backbone of safety work is supervision, and the development of employees into specialists who become familiar with the hazards of their occupation and what may be done to overcome them.

Good housekeeping has doubtless played a part in making this record possible, as the mine both on the surface and underground is kept scrupulously clean and orderly. Many of the men in the mine and about the plant have received first-aid training from the United States Bureau of Mines, and they have been kept familiar with first-aid work by company instructors. A well-equipped first-aid station is maintained underground; on the outer wall of this room and along the main entry is a lighted bulletin board on which are displayed posters of the National Safety Council and general plant notices or rules. The plant doctor visits the property daily and inspects all first-aid cases in the plant and in the mine.

No printed safety rules have been issued, but the foremen instruct employees in such rules as are formulated at the regular monthly foremen's safety meetings. A foremen's safety code which covers the responsibility of such men has been prepared by the company for the guidance of its foremen. The general safety committee is divided into five subcommittees--the inspection, fire, first aid and sanitary, publicity, and educational committees; and, if necessary, an investigating committee is appointed.

According to a paper written by the superintendent, F. C. Brownstead, the success of accident prevention at the entire Ironton plant is attributable to education and cooperation. It has been found by this organization that the foremen are the logical persons to have on a safety committee; in fact, little progress was made until the responsibility for accidents was placed on the foremen. After a shutdown and before resumption of work all the men are called together and urged to be careful. Eye protection is aided by the compulsory wearing of goggles when the nature of the work requires them, because prior to 1925 many accidents were caused by flying chips of stone and steel; each man is charged $1.10 a pair, but the company keeps the goggles in repair free and refunds the money when the employee leaves. The rule requiring men to have goggles with them at all times eliminates any excuse for an eye injury. Apparently it has also eliminated eye injuries.

Every new man is given a thorough physical examination. If a man receives a slight injury which may handicap him on his regular job he is transferred to some other job temporarily, but no job is created for him. The cooperation of the company in accident prevention favorably impresses the men. Employees of the Alpha Portland Cement Co. receive a cash bonus of 1 per cent of their earnings for each month their plant operates without a lost-time accident. Because of its outstanding record the cash bonus of the Ironton plant was increased to 2 per cent in 1930 and is still being paid. The average monthly bonus for 31 consecutive months was $1.25 and is a form of recognition which is appreciated.

This company has had an outstanding accident experience--not a lost-time accident in the entire plant since December 8, 1926. On January 1, 1932, the entire plant had operated 1,849 days without a lost-time accident; the no-lost-time accident record at that time was exceeded by only one other plant--namely, the Lehigh Portland Cement Co.'s plant at Iola, Kans., which on January 1, 1932, had operated 1,939 days without a lost-time accident. Since January 1, 1931, to date (September 26, 1932) the Ironton plant has operated 289 additional days, or a total of 2,138 days, without accident.

In 1928, the mine at Ironton won the "Sentinels of Safety" trophy for nonmetallic mineral mines in the National Safety Competition of that year, and on January 1, 1931, the Alpha Company was awarded a Joseph A. Holmes Safety Association certificate, at that time having worked its entire plant 1,484 days without a lost-time accident. In addition, the Ironton plant has won the Portland Cement Association trophy five consecutive times for having operated an entire calendar year without a lost-time accident.

Unquestionably, the very fine safety performance of both the underground mine and the entire plant is due in large part to the excellence of the supervision; the fact that the underground mining work has been conducted for six years without a lost-time accident, even though such work is normally held to be extra hazardous and to be inherently subject to accidents of both minor and major character, indicates that when efficient industrial plant safety methods are applied intelligently to underground mining work, essentially the same excellent safety results can be achieved as are now being obtained rather generally in well-operated industrial plants.

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UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

METAL-MINE FIRES AND VENTILATION

BY

D. Harrington

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DEPARTMENT OF COMMERCE - BUREAU OF MINES

METAL-MINE FIRES AND VENTILATION[1]

By D. Harrington[2]

Metal-mine fires in the United States are not of frequent occurrence and relatively few of those that do occur result in loss of life, although many cause heavy property loss or are costly to extinguish. Over a span of years, however, more lives are lost in metal mines from fires than from any other of the untoward occurrences which afflict metal mining, such as electrical storms, floods, cave-ins, magazine explosions, or strikes. In perhaps more than 90 per cent of the fires in metal mines there is no loss of life; hence, the general public, even the general mining public, seldom hears of these fires, though the property loss frequently reaches hundreds of thousands of dollars; on the other hand, in some instances the property damage from an underground metal-mine fire has been negligible and the loss of life comparatively high.

In any comprehensive study of metal-mine fires, ventilation is certain to be found a vital factor. The influence of ventilation on fires and the influence of fires on ventilation are of vital importance. In metal-mine fires very few persons meet death from actual contact with heat or flame; nearly all fatalities are due to asphyxiation or suffocation. A well-planned, well-installed, and well-operated mechanical ventilation system is unquestionably one of the best available present-day protections from loss of life and property in case of fire in a metal mine; that this is true is attested by the usual helplessness of mining officials in trying to handle underground fires in a mine that has not had the foresight to install mechanical equipment to control underground air flow. On the other hand, the establishment of high-velocity air currents without sufficient day by day attention to such details as the method of coursing, or controlling, or confining them most certainly tends to increase dangers from fires rather than to aid in the avoidance or elimination of these dangers; the truth of this latter statement is shown by the fact that the greatest loss of life in metal-mine fires during the past 20 years has been in relatively well-ventilated workings, and in every case the life loss has been due to some more or less trivial or easily remedied defect in the ventilating system - usually a defect that has had some reference to doors used or supposed to be used to deflect or control air currents.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6678."

2 Read before the Mining Section, National Safety Council, Washington, D. C., October 4, 1932.

2 Chief engineer, safety division, U. S. Bureau of Mines.

In a study of loss of life in metal-mine fires in the United States it appears that in the period 1916-1922, inclusive, approximately 240 lives were lost through fires, but since 1922 only approximately 20 persons have been killed from that cause. As far as can be ascertained from available records, approximately 600 lives have been lost in metal-mine fires in the United States to date. This does not appear to be a very impressive total until one realizes that in one of these fires 163 lives were snuffed out, in another 47, and in another 21, and that these three fires with relatively heavy individual life loss all have occurred within the past 17 years. In addition, the Hollinger gold mine in Ontario with underground operating conditions and practices similar to those of metal mines in the United States lost 39 lives in an underground fire in 1928, keeping to the front the fact that our metal-mining people still have a heavy hazard from fires even though they have caused the loss of relatively few lives in the United States since 1922.

Few mining companies, coal or metal, care to divulge the facts connected with fires or with any kind of accidents at their properties; hence, available information is meager, not only as to the number of fires but also the conditions which caused the fires or under which they were extinguished or otherwise controlled. The forces of the safety division of the Bureau of Mines for over 20 years have been on the alert to obtain data about all kinds of accidents in mining, and these include fires or explosions. During recent years the field men of the safety division have annually been present at or shortly afterwards visited the scene of about 35 explosions and 25 fires in mines, a few of the explosions and about one-fourth of the fires being at metal mines; however, there is good reason to believe that at least 100 fires occur annually in the metal mines of the United States when these mines are operating at even moderate capacity.

Unquestionably the most dangerous aspect of the fire hazard at metal mines is the far too prevalent idea among metal-mining people that their particular mine is not subject to fire, that there is virtually nothing in the mine which could burn, that the product mined is incombustible, or the mine is too wet for a fire to get a start; these and dozens of other equally falacious reasons are given for not expending any considerable amount of time or funds or effort in safeguarding against fires or for not providing for the handling of one if it should occur. As a matter of incontestible fact there are few underground mines in the United States which are not subject to numerous fire hazards; practically all underground mines (coal, metal, or nonmetallic) have combustible material in them such as timber, explosives, oil or gasoline, insulated electrical wiring or equipment, or similar materials the burning of very small quantities of which underground can easily result in many deaths; and in addition some metal mines have combustible shales or pyritic ores which may fire spontaneously, or have explosive gas, and far too many have flammable surface structures or heavy growths of dry grass or shrubs so near the entrance that if fired they would be likely to send

poisonous gases into the mine and endanger the lives of all underground. In addition to the fact that practically all underground mines have in them ample combustible material to cause heavy loss of life in case of fire, practically all underground metal or nonmetallic mineral mines have in them numerous possible sources of ignition such as open lights and smoking, open-arcing types of electrical apparatus, and, in some mines, gasoline or other internal-combustion engines, blow torches, and other devices or equipment which give off flames.

In a study of the source of more than 100 metal-mine fires which occurred in the United States prior to 1922 it was found that about 35 per cent were caused by ignition of timber by candles, about 10 per cent were from using heat to warm oil or explosives or for some similar purpose, about 10 per cent from flame or gases sent underground from burning surface structures, about 12 per cent from spontaneous combustion (from timber splinters or bark, stored hay or manure, carbonaceous shales, pyritic ores, oily waste against timber, etc.), about 7 per cent from electricity, about 3 per cent from explosives, about 3 per cent from carbide lights against timber, and about 20 per cent from unknown or miscellaneous causes including incendiarism, lightning, etc. In the foregoing it will be noted that open flames - including lighted candles, carbide and oil lamps, and fires for heating and other purposes - were responsible for at least 58 per cent of these underground fires, electricity for but about 7 per cent, and that spontaneous combustion, explosives, and unknown causes accounted for the other 35 per cent.

In a study of a considerable number of metal-mine fires in a certain very active and progressive metal-mining section of the United States in which scores of fires had occurred, it was found that those which were dated before 1917 were caused about as follows: 50 per cent by candles and oil lamps, 2 per cent by carbide lights, 10 per cent by heating or other fires, 4 per cent by electricity, 4 per cent by smoking, and the remaining 30 per cent by unknown or miscellaneous agencies such as lightning, use of explosives, spontaneous combustion, friction, etc. Here again open flames of various kinds caused at least 66 per cent of the fires, while electricity caused but 4 per cent. Data were at hand on a larger number of mine fires in this region subsequent to 1917 than those above described for the period up to 1917, and the causes were essentially as follows: Electricity 40 per cent, carbide lamps 10 per cent, open fires for heating and other purposes 12 per cent, smoking 11 per cent, and welding torches 4 per cent; the remaining 23 per cent were unknown and miscellaneous causes such as spontaneous combustion, lightning, friction, etc. The above data cover more than 100 metal-mine fires in one metal-mining region with conditions which might be interpreted as against rather than in favor of easy or frequent occurrence of fires; the cost of fire fighting has been enormous in this region, though loss of life has been relatively slight. It is significant that with the virtual disappearance of the candle from the mines about 1916, the percentage of fires caused by the miner's light dropped from about 52 per cent in the period before 1917 to about 10 per cent in the period subsequent to 1917, though the percentage caused by carbide lights rose from 2 per cent for the period before 1917 to 10 per cent for that after 1917.

Perhaps, however, the most significant trend shown in these figures is that indicating that fires of electrical origin rose from about 4 per cent of those which occurred before 1917 to 40 per cent (or a tenfold increase) for the years 1917 to date; here it may be well to state that a considerably greater number of fires were studied in the latter than in the former period.

An attempt was made in one study on more than 100 metal-mine fires to locate the particular part of the mine in which the fires started. It was found that about 20 per cent started in the shaft and that in some cases the shaft was damp to wet; slightly less than 20 per cent started in a shaft, pump, or similar underground station and in at least half of the cases these places were damp to very wet; about 25 per cent started in timbered drifts or crosscuts; about 12 per cent in stopes; about 10 per cent in ore or waste chutes; about 3 per cent in raises; about 3 per cent in explosives magazines; and about 3 per cent in surface structures; the remaining 4 per cent had their origin in miscellaneous places such as mule barns, caved regions, etc. The fact that this particular study in which over 100 fires were involved showed that about 40 per cent were started in shafts or in shaft or pump stations gives a definite indication as to the advisable points at which the most definite fire preventive precautions should be taken; it is also significant that approximately 65 per cent of the fires in this region started in shafts, shaft or other stations, and drifts and crosscuts, all of which should be readily accessible for inspection purposes as well as for the application of fire-prevention installations and practices; in other words, it would appear that inspection and supervision were inadequate in the mines that had these fires.

The latest available data indicate that between 30 and 40 per cent of present-day metal-mine fires owe their origin to electricity. This percentage is likely to increase with the widening use of electric locomotives, pumps, and other equipment, as well as with the rapid introduction of electrically operated hoists, blower fans, slushers, drills and like equipment into stopes and other face regions in the mechanization of our mines. This would indicate that utmost care should be exercised in the selection, installation, and use of electricity underground, and certainly all electrical installations should be kept under the closest scrutiny and in the best of repair at all times. It seems also to be established that between 30 and 40 per cent of present-day fires in metal mines are due to smoking and to the use of carbide lights and other open fires or flames for various purposes; this seems utterly needless, and it is inconceivable that our metal-mining people should continue to take the "long chances" of the probable heavy loss of life or at least of property by fires started by open-light lamps, smoking, or use of open flame for any reason, where there are readily available far safer and far more efficient portable electric lamps for miners, smoking is utterly unnecessary and utterly reprehensible in a mine, and there is little or no necessity for having or for allowing open flame of any kind in any mine for heating, thawing, or for any other purpose except that possibly under some conditions and for limited periods

a blow torch or an electric or acetylene welding apparatus may be used safely if reasonable precautions are taken. Evidently at least three-fourths of our metal-mine fires are now started by electricity, open lights, smoking, and flame used for heating or other purposes underground; this fact should indicate what precautions may be taken to avoid most of the present-day fires in metal mines. Many fires start just after the working shift leaves the mine; hence, it is important to have a fire patrol during nonworking periods and at least for the first hour or two after the working shift.

Below is a brief synopsis of the cause of several relatively recent metal-mine fires:

1. Some one, either on one of the levels or when coming up in the cage, threw a lighted cigarette butt away, and this started a fire.

2. Timber in an underground shaft gave way, breaking the electric wires that were in the shaft; as the shaft was very dry, the timber was ignited by the wires.

3. A miner or some other person going off shift threw a lighted piece of paper down a chute to see how much ore was in the chute. A fire resulted.

4. A fire was started close to the fan motor on one side of a shaft station by contact between a miner's carbide cap lamp and canvas tubing taking air from the fan to the raise where miners were at work.

5. A fire was caused by contact of an overheated incandescent electric-light globe with some "dope" used for a belt.

6. A match, presumably thrown by a smoker, started a fire in a station close to a timber shaft.

7. A fire was caused by a spark from a short circuit in the hinge on a knife-type trolley switch.

8. A pumpman, burning newspapers on a concrete floor in a pump room, set fire to adjacent timber in a mine.

9. The bark on a timber cap piece was fired, probably by coming in contact with a lighted carbide lamp on a miner's cap.

10. A candle used for illumination was left burning by a miner and caused fire.

11. The contact of a trolley wire with the cap of a sagging set of timber in a wet drift, which was being crushed, caused a smoldering fire which gave off considerable volumes of dense smoke.

12. Carbonaceous shale which throws off methane was encountered in a shaft sunk in driving a tunnel for the water system in a big city. The shaft was so wet that the workers had to use rubber boots, slickers, rubber caps, and hats. The methane was ignited by an open light and the resulting flames came in contact with timber; notwithstanding the fact that water was pouring in the shaft, the timber was ignited and burned briskly.

13. Fire was started by the contact of electric wires with 12 by 12 inch timber, probably due to defective insulation.

14. A small transformer and a starting compensator ran hot, setting fire to a wooden frame, which was burning when the fire was discovered.

15. Men were using an acetylene torch in a pocket of the shaft station and went away. Later on fire was found.

16. One report said: "It did not seem possible that a fire could occur in this shaft. The timber burned, and the mine is wet." The fire was caused by a short circuit in the electric-power cable.

17. A fire was caused by an overheated resistance on one of the electric tuggers used for slushing ore.

18. Fire was found in a raise 80 feet below the 300-foot level at a point where spent carbide was thrown.

19. Lighted powder wrappings thrown into a chute in ascertaining how much dirt was left caused a fire.

20. Sparks from a railroad locomotive set fire to the wooden headframe on the surface. Hot embers from the headframe fell into the shaft and burned it out.

21. The open flame of the lamp of a man who was recovering some tools that he had hid in a place where there was some dry lagging probably came in contact with timber and was the cause of a costly shaft fire.

22. It was determined that a shaft fire was started by a carelessly thrown cigarette, although there was a "no smoking" rule in the mine at the time. Very frequently the "no smoking" rule supposedly in effect in and around mines is most flagrantly violated by the mine foreman or the shift boss.

23. Fire was caused when the rope on a slope ran against the bearing on a roller and heated the oil that was in the roller bearing. The burning oil set fire to the nearby ties, causing a costly and destructive fire.

24. A lighted carbide lamp which a powderman placed on a shelf under a large fuse box started a fire by igniting the paper between the boards.

25. A trolley wire had fallen, and not being sufficiently grounded to operate the circuit breaker at the shaft, it heated, caused the insulation on the feeder wire to burn, and started a fire in the shaft. In a number of instances trolley wires have been down, set fire to cables, or set fire to cable insulation which carried the flame some distance to timber and caused fires.

26. A match or burning tobacco from a pipe started a fire.

27. Current was left on the controller of a sinking hoist; the hoist motor became overheated and the timber foundation began to burn. (Why have electrical machinery on timber foundations? It is almost universal practice for temporary installations underground, but should not be allowed.)

28. Blasting of cribs in top-slicing work or in cut-and-fill work or blasting of posts where the timber mat is following has caused many severe and costly mine fires, although they have not resulted in very great loss of life.

29. Some posts which were being blasted out to bring down the overlying material in a mine using a caving system caught fire, undoubtedly from the fuse, and before the fire was brought under control the entire mine had to be flooded, entailing an enormous loss to the company. The cause is apparent, though the answer is not quite so easy. The use of permissible explosives is recommended for blasting of timber; and where it can possibly be done, the blasting of timber electrically rather than by the use of fuse is advisable.

30. In some mines fires have been started by dynamite used in large charges in high sulphide ore (in which sulphur content is 30 per cent or over); in some cases there have been explosions very much like those which occur in coal mines.

31. The wiring of an electrically operated blower fan supplying air to the face of an exploration drift started a fire because of squeezing ground with short-circuiting of the wires.

32. An oil switch controlling the power lines of a metal mine in some manner "failed" and started a fire at the shaft collar on the surface. The flames spread into the mine and caused severe property loss.

Timber is by far the most widespread combustible in metal mines, and under certain conditions will burn whether in place in workings or only piled ready for use; whether wet or dry; whether in form of crushed decaying mat or in solid posts, caps, or girts; or whether in an absolutely open space or in an abandoned back-filled stope. A remarkably small amount of burning timber,

such for instance as four or five timber sets, can give off a sufficiently large quantity of poisonous fumes to kill scores of people. Sulphide ores in a finely divided state burn readily, especially if in contact with burning timber; and copper and iron sulphides appear to burn much more readily than lead or zinc sulphides. A most difficult fire encountered in metal mines is one in a timbered stope back filled with waste-rock material, especially that containing considerable percentages of finely divided copper or iron sulphides. Other flammable materials which have aided in starting or extending metal-mine fires are gasoline, oil, and grease kept in open places underground.

Leaving piles of bark, chips or shavings of timbers in exposed places, or throwing empty explosive boxes, excelsior, sawdust, manure, spoiled hay, oily waste, old clothing, and refuse into abandoned workings - all these careless practices have resulted in starting or in aiding the extension of destructive fires.

At all mines that have flammable surface structures near the entrances, fire doors should be provided to prevent fumes or fire from being drawn into the mines. Flammable surface structures should not be closer to a mine opening than 100 feet, but if already at or near the mine opening such structures should be fireproofed by gunite or some other method and thoroughly protected by readily available water lines such as hose, sprinkler systems, and fire extinguishers, the equipment to be kept in good working condition by periodical testing. Combustibles should as far as possible be kept out of structures near mine openings, and the use of fires, matches, and smoking materials in mines or in flammable structures near the openings to mines should be prohibited or at least very definitely restricted.

Intake-air shafts are generally dry and if timbered constitute a continual fire hazard. It is noteworthy that a large proportion of fires in metal mines originate at or near shafts. Downcast shafts should be of concrete construction, and where this is not feasible timbered shafts and shaft stations should be fireproofed by gunite or other means. In the fireproofing process it is desirable to make the shaft lining smooth, as this greatly facilitates the flow of air.

In fireproofing by gunite it is advisable at intervals of approximately a hundred feet (preferably just above stations) to take out about one set of timbers around the shaft perimeter and concrete back to the rock walls to prevent creep of fire in timber behind the gunite.

Where it is not deemed feasible to fireproof downcast timbered shafts by concreting or guniting or other effective method, the shafts should be thoroughly protected by sprays, water pipe with fusible plugs, or perforated pipe, and with suitable placed valves, so that in case of fire in the shaft any section may readily be drenched with water. Preferably, the water lines should be controlled from the surface, and there should be suitable pressure-reduction valves and signboards denoting the exact location of control valves. There should be on important shaft stations at least 50 feet of fire hose

connected to the water line, with nozzle attached and with valve controls near at hand (well designated by signboards), in order that water may be always available for use without delay at the stations or in the shafts. Care should be taken that water used in shaft fires should be so applied as not to reverse air currents and thus possibly cause loss of life. Where the ventilating current is weak, an upcast air current can be reversed to downcast by the spraying of a relatively small amount of water into a shaft.

Timber upcast shafts which are usually damp or wet, or downcast shafts in which some or all of the timber is damp, may burn readily if fire gets a good start in a dry, heavily timbered station or other adjacent working. Hence, fire protection and fire prevention should not be neglected, even in upcast shafts, or where the timber in a shaft or station is usually damp.

Since many fires in timbered shafts are started by electric wires, it is desirable, where feasible, to transmit electric power underground through drill holes; this is done satisfactorily in several places. Where power wires must be taken through dry downcast shafts, it is advisable to fireproof at least that part of the shaft containing the power wires and in any event such power wires should be very carefully installed to prevent leakage or short-circuiting of the current.

Electric trolley wires should not be allowed to be in contact with timber caps; wires carrying current should not be attached to timbers by nails, pieces of rope, and wires; electric motors should not (if it be possible to prevent it) be placed close to timbers or in heavily timbered stations or under heavily timbered stopes or raises, and if they must be in such places the immediate surroundings should be fireproofed (by gunite, concrete slabs, or otherwise). Electric-light globes should not be in contact with timber; fuses should not be bridged by heavy wires; circuit breakers or no-voltage release compensators should not be locked in place; and the usual slip-shod methods of installing mine electrical equipment and of making repairs on it should be supplanted by work of a nature at least as careful as that required for surface electrical installations. Suitable fire extinguishers or containers with sand should be readily available at or near all electrical stations in a mine and preferably just outside the station and on the intake or fresh air side of it.

Electric switches should be of the inclosed externally operated type of substantial and safe construction, attached to fireproof base; and the region around the base, as well as the region around the wires near the motor, should be fireproofed. Switches suitably designated by colored light or by signboard should be provided at reasonable intervals on each level so that current to electrical equipment or power lines may be cut off without delay; and electric current should be excluded from mines or parts of mines which are not being worked or kept under more or less constant supervision. Trolley locomotives should be superseded where possible by storage-battery locomotives. Where trolley wires are guarded by wooden

trough, occasional sections of the trough should be made of noncombustible material, as fires have been transmitted several hundred feet along trolley guards in a drift otherwise devoid of timber.

Storage of even small quantities of oil, grease, gasoline, or similar flammable materials underground is dangerous, and if necessary should be done under lock and key in a fireproofed place isolated from travel way and air currents. These dangerous substances should not be allowed to lie around timbered shaft stations or other workings, as is common practice.

Similarly, explosives should be stored underground in a place isolated from ordinary travel and air currents, the "caps" and fuse at least 100 feet distant from the "powder," and should be issued by an attendant only at certain stated periods. These places should be kept clear of excelsior, paper, empty boxes, parts of boxes, and other refuse. All open lights should be absolutely excluded, the place being lighted either by the ordinary incandescent electric lights properly installed or by small "battery" lights. It is advisable to have fire patrol of working places immediately after blasting, especially where blasting is done in or close to timber. Where feasible, blasting should be done electrically rather than with fuse. In blasting timber, not more than five or six holes should be fired simulatenously at the same place, and it is advisable to use the coal miner's permissible explosives to decrease the fire risk. The greatest care should be taken in transporting explosives in bulk to the underground magazines and also in smaller quantities from the magazines to the working place.

Candles should be entirely excluded from mines except for the purpose of occasional testing of oxygen content of air; and the metal miner should learn to carry his carbide lamp in his cap as the coal miner does, for the use of the "candlestick" in holding a carbide lamp to timber post has been responsible for numerous fires. Notwithstanding the opposition to it, a "no-smoking" rule should be instituted and rigidly enforced in metal mines, and bosses as well as other underground employees should obey it. The present-day electric cap lamp is more efficient, less expensive, and far less hazardous than the acetylene lamp and by all means the metal miner should discard the open lights and use only up-to-date electric cap lamps for his portable lighting.

As every metal mine which has timber or other flammable material underground has a fire hazard, every such mine should take precaution to combat fire, and one of the most essential precautions is to have available - not only in shafts and shaft stations, but on important levels - water lines at least 2 inches in diameter and with suitable surface storage and valve control. The exposed threads on valves or water pipe in mines should be kept well greased or oiled to prevent undue corrosion; this is an important detail and one almost universally neglected. The practice of converting air lines to water lines is good, but care must be taken that locations of valves to accomplish the change are well marked, with full instructions by signboards at or near the valves. In deep mines where the head would be excessive, suitable pressure-reducing valves or other equipment should be introduced into lines or branches for attachment of fire hose, to hold pressure to less than 100 pounds per square inch at the hose.

If a fire can be directly and promptly attacked by appropriate means, it generally can be extinguished before much damage is done; hence, it is important to have available at critical points - important shaft or other stations - emergency equipment such as fire hose and extinguishers or a supply of sand or similar material. Of course, it is absolutely essential that such equipment or material be kept in working order by periodical try-out or rigid inspection and by the instruction of the men in its purpose and use.

There should be at least two openings through either of which men may escape without danger or difficulty when one is not available temporarily; if the mine is deep (say, 500 feet or over) and employs a considerable number of men (say 25 or more underground on one shift), two shafts should be available, each with hoisting equipment to the lowest levels, able to remove the men from the mine with minimum delay.

There should be more than one system of shaft signals, and if feasible, there should be a system of signaling from the moving cage. Telephones should be installed on important underground shaft stations and at important gathering places in the mine, and if feasible the telephone lines should be brought into the mine through some opening other than the one ordinarily used for handling men. Where it is feasible, air and water lines should be brought into the mine through two openings, to be available from one in case the lines are destroyed in the other.

Signboards, always kept in good condition, should be placed at comparatively numerous points underground to designate the direction to exits as well as to denote places of danger. New employees should be promptly acquainted by bosses with the location and best methods of reaching the various exits and should be required to use these exits at least once every few months, and the travel ways in these exits should be kept in safe and "travelable" condition.

There should be installed some system by which miners may be promptly notified of fire, and all employees should be informed as to the significance of such signal. Turning compressed air on and off, filling compressed-air lines with water, rapping on pipes, ringing of gongs, flashing of lights, use of telephones, and introducing some stench into compressed-air lines are the methods in use; the latter is one of the most efficient.

Probably the best and most effective method of protecting both life and property in case of fire is the establishment and maintenance of a carefully designed, efficient, mechanically controlled ventilation system with (1) air-tight doors so placed as to be able to isolate shafts from mine levels; (2) air splits held absolutely separate from each other - hence, able to confine smoke to but a small part of the mine; and (3) the main man way downcast or on the intake - hence, allowing escape in fresh air unless the fire is in the main travel way. Positive control of air currents at all times is essential, as even small defects in ventilation arrangements may be costly.

Leaky doors which only partly hold air at ordinary times and fire fumes in time of emergency, open levels between main intake and return shafts, or open raises of crosscuts between separate splits - and, in fact, anything which allows short-circuiting of air - is likely to be dangerous. Even the trolley slot in doors or doorways should be made as small as possible.

The following additional suggestions are made as to mine doors:

1. Mine doors and their frames should be substantially constructed, should be covered with roofing iron or otherwise fireproofed, should be tight, and should be always kept in repair.

2. Doors should close automatically after passage of men or cars and should have a latch or other positive means of keeping them closed, even though ordinary pressure on the outside of the door should become pressure on the inside, a situation likely to occur at time of fire.

3. Every main air course, whether shaft, incline or level, should have in place either within itself or in all the openings leading from it, a system of doors such that in case of emergency the entire shaft, incline, or level opening can be quickly isolated from other parts of the mine. This is especially important as to downcast, or intake timbered shafts, inclines, or level air courses. Doors should be provided for all main openings leading from main air courses (both intake and return); and when at ordinary times there should be free flow of air, the door should have a sign denoting that it is to be closed only in case of emergency (if possible giving the exact contingency under which the door should be closed).

4. All doors, but especially those for use during fires or in exits between mines, should close flush against the face of the doorframe instead of in a groove within the frame; this is to prevent sticking of door or inability to open or close it if there should be deformation of doorframe from ground movement or swelling of the door or frame.

5. Where there is necessity for frequent opening of doors controlling main air currents, or where such doors must remain open for long periods, it is advisable to have two or more doors placed so that when one must open, one or more may be closed and thus prevent passing of air.

6. All employees should be educated to respect doors and especially to close those which are supposed to be closed. It is advisable to have on each door a sign such as "Keep this door closed," or "Close this door in case of fire in main shaft," or other sign applicable to the specific condition existing at each door. Any employee failing to close a door which should be kept closed, or one who deliberately or carelessly breaks a door or its frame, should be disciplined, even to the extent of discharging him, if necessary.

7. It is extremely desirable that all mines be divided into air splits by doors, etc., in such a way that a fire in any one split will be held separate from all other splits and by manipulation of doors may be further isolated and controlled. In ordinary times, this system operates to the benefit of distribution of air currents to the places where men work.

Ventilating one mine through another is not advisable if other circuits are possible, for experience has shown that a fire in one mine may overcome or kill men in an adjacent connected mine. The mine which ordinarily delivers air to its neighbor may at time of fire receive deadly fumes through reversal of air currents. This danger from interventilation of mines is present whether the mines are owned by separate companies or by one company, and it exists to a certain extent where ventilation is mechanically controlled, though not nearly so much as where the ventilation is natural. Where two mines have been stoped along the boundary line, in some cases slime or sand filling may be used to make barrier pillars.

The main ventilating fan should be in a fireproof housing and, if possible, be placed on the surface and connected to mine openings by fireproof ducts. It should be equipped with a system of doors, at or near the fan, or with any other feasible method which will allow of changing direction of air currents with minimum delay. However, at time of fire, the ordinary direction of underground air currents should be reversed only after mature consideration of the effect upon those underground. The following hints are offered in connection with reversible fans and reversing of air currents:

1. All main fan installations at metal mines with fans on the surface should have the reversible feature, and where feasible, underground main fans should also have the same arrangement.

2. In addition to the reversing feature, all fan installations should be fireproof; and main air courses should be so equipped with doors that, if necessary, all air flow may be closed to or from the air course.

3. To make the reversing feature effective, mine doors should be so constructed as to remain closed, preferably by a positive latch, even when the direction of pressure against the door changes. Occasionally in normal times the reversing feature should be tested to make certain of its smooth working in case of emergency.

4. Booster or distributing fans in use in metal mines to distribute air to "blind ends" need not be reversible, but should be in a fireproof setting.

5. While there are instances of loss of life due to reversal of air currents, there are a greater number of instances of saving of both life and of property by use of reversing systems in both coal and metal mines.

6. Danger in reversing direction of air flow is much greater in coal mines than in metal mines, owing to the possibility of explosions, yet sentiment in coal mines is almost unanimous in favor of installing the reversing feature on main fans.

7. It is no more logical to condemn the use of the reversing feature in connection with main fans at metal mines because of one or two isolated cases of loss of life associated with reversal of air after a disaster than it is to condemn use of hoists for hoisting men because there have been a few hoisting accidents, or to condemn the use of electricity because there have been some deaths in mines from shock.

8. The cost of installing main fans with the reversing feature is but about 25 per cent greater than without it. The total cost for a fairly large mine would be less than $2,500, and for an ordinary mine not as much as $1,000. With use of this feature, the return in dollars and cents would probably be many times the costs just cited, and the value returned in saving of life might be incalculable.

9. The reversing feature can be used to facilitate ordinary mining operations. One deep metal mine in the West uses the feature to "cut" ice out of the operating shaft in winter, and converts the downcast operating shaft to upcast in less than 10 minutes.

10. Any metal mine, which for ordinary operations must for some reason maintain a main shaft or travelway as upcast, should certainly have the reversing feature on the fan, since a fire in almost any part of the mine may at any time fill the shaft with fumes and quick reversal of the direction of air flow would give underground men a safe exit through the shaft. Such shafts should also have doors at all intersections so located and constructed as to be able to isolate the shaft from all air flow if necessary.

11. Reversing the direction of air currents unquestionably has some possible dangers, some phases of which are herewith mentioned: (a) At time of fire or explosion, changes in direction of air flow should not be made except when it is fairly well established that good rather than harm will result.

(b) Mines with fans equipped to reverse direction of air currents seldom use this equipment; hence, at time of emergency there is likely to be undue delay in effecting the change, and in some cases the change can not be brought about due to inefficiencies in the installation.

(c) Mines having fans with the reversing feature rarely have underground doors so constructed that the reversal can be accomplished.

(d) When air currents are reversed in mines with high humidity and acid water they may cause damage to metal in downcast shafts when they are converted to upcast.

(e) Mines with such fungus and timber decay in abandoned and partly ventilated workings tend to spread fungus spores; hence, in such mines timber decay is extended by reversing air currents.

12. In general, very few persons care to assume responsibility for changing air flow in a mine at time of disaster; moreover, very few have the special knowledge as to the feasible method of making the change. These conditions alone will, in general, safeguard the reversing feature from misuse.

13. The direction of main air currents should not be changed except under instructions from the mine foreman, superintendent, or other official who is familiar with the mine and its air currents, and who should also be sufficiently familiar with underground conditions at the time of making the change that he realizes the probable effect and feels able to shoulder the responsibility.

14. The placing of the reversing feature on main fans for metal mines is a cheap and effective method of providing insurance against loss of life and property in case of fire. It pays even if never used, and it is likely to yield big "dividends" in prevention of losses in case it is available and is used in an emergency.

Comparatively few men at metal mines understand methods of inducing or controlling the flow of air at ordinary times, and still fewer know how to handle the gases from a fire. It is a fatal mistake to assume during a mine fire that a man is safe in clear air so long as his carbide light burns. Men who were in air devoid of smoke have been found dead from carbon monoxide with their carbide lights burning brightly beside them. The candle or carbide lights burn brightly in air containing enough carbon monoxide to cause death in a fraction of a minute; however, a man can live in air containing too little oxygen to support the flame of a candle or a carbide light. A candle is extinguished when the amount of oxygen falls below 17 per cent, and the carbide light "goes out" when the oxygen is below 14 per cent; a man can exist for some time when the oxygen content of the air falls as low as 10 per cent, if no other harmful gases are present. A canary or a mouse is quickly made unconscious by a percentage of carbon monoxide in which a man can live for a short time; hence, when the canary drops, the man should take warning and retreat immediately. When venturing into air suspected of being impure, if the small animal (canary or mouse) lives and a candle or carbide light burns, a man is safe unless explosive gas is present, and this method of testing should be adopted in metal mines even in face of the ridicule usually offered when the animal is used.

When men are unable to escape to the surface at time of mine fire, they generally can aid materially toward saving themselves if they do not lose their heads by attempting to force their way through the smoke and gases.

By retreating to blind workings into which the smoke has not penetrated and tightly sealing off the openings to exclude the gases, using timber, posts, boards, plank, canvas, paper, mud, dirt, or even clothing to make the seal, men may live for days, especially if they have included within the sealed region one, two, or several hundred linear feet of fresh-air drift, have taken with them a supply of water and such food as they may have, use food, water, and light sparingly, and move about occasionally to keep the air within the sealed place properly mixed. The moving about is important, as otherwise the air may be locally depleted of oxygen where many men sit or lie still, and they may die for lack of oxygen, although plenty may be readily available 20 or 30 feet distant. In selecting a place to barricade, care must be taken to choose solid ground, as barricades placed in filled or broken ground are likely to admit gases, and at several fires have done so, with the resultant death of the men. Similarly, care must be taken to seal all openings and to keep the seals tight; in at least one instance men tried to seal themselves, but apparently did not see a raise; later, fumes entered through the raise and suffocated them.

Some mining companies now have underground refuge chambers with air-tight doors; in these chambers are placed barrels of water, compressed-air lines, compressed foods, and telephones. It is suggested that mines with bad fire hazards be equipped with such stations. Mines with only one opening to the surface should be required to have such refuge chambers for use in case of cave-ins and fires in the one opening.

Only too frequently a fire obtains so great a start that it is impossible to handle it directly, or it is in an inaccessible place, or in a place in which falling material prevents actual contact, and recourse must be had to other methods of attack. To control such fires, the most successful and acceptable method is to place seals of clay, sand, boards, canvas, or, preferably, concrete in such manner as to confine the fumes within a comparatively restricted area with minimum leakage of air into the region. As a rule this can be done only after much dangerous effort. Where the seals are made tight and kept so by constant watchfulness, and where the surrounding strata are not broken, the oxygen content of the air is soon depleted below the 5 or 6 per cent necessary to support combustion. If the region is then allowed sufficient time to cool before being opened, fires are generally controlled with minimum expense, though the cost may mount into the thousands or even into the hundreds of thousands of dollars.

Where an entire mine must be sealed or where territory with much open space (say 1,000,000 cubic feet or over) must be sealed, the process of extinguishing a fire by sealing is likely to be long drawn out and may not be feasible. This is especially true when any opportunity for air leakage exists, as through intersecting veins or connecting workings to other mines, or breaks or openings of any kind to the surface. In such circumstances other methods of control must be utilized, though in any event sealing is likely to constitute an important part of the process.

The usual method of overcoming a metal-mine fire where sealing is not effective is to flood it. This method is costly, both in actual performance and in subsequent effects. Sometimes steam is tried for extinguishing a fire, but generally with only limited success. At other fires some inert gas, such as carbon dioxide, is used. Water sometimes is introduced through diamond-drill holes and coursed through the area. Some fires of long standing have been overcome by the introduction of hundreds of thousands of tons of mill tailings through diamond-drill holes and other openings; this process usually accomplishes desired results, after sealing and other available methods have proved unsuccessful.

The cement gun is extremely useful in cementing broken ground to prevent escape of fumes, and also in making bulkheads of timber, rock, and boards air-tight, and even in constructing cement bulkheads.

Where fire is in moving or in heavy ground, rigid bulkheads of cement and rock are inefficient, and bulkheads of timber laid skin to skin can be held tight much more readily than cement and rock; with cement and rock bulkheads it is frequently advisable to construct the upper part and possibly the sides with clay to allow for movement without cracking the bulkhead.

All bosses, surface and underground, as well as clerks, timekeepers, and other employees, should be brought together into an organization to discuss possible fires from time to time, and to determine definitely what each should do in case of fire on the surface or at various places underground. There should be posted in the mine office and kept up-to-date a list of names and street and telephone addresses of persons to be notified in case of fire, including mine officials (manager, superintendent, and foreman), safety men, oxygen mine rescue apparatus men, state inspectors, doctors, and ambulance drivers. All bosses, cagers, motormen, drivers, and samplers should be made fully acquainted with location of valves and electrical switches and should be thoroughly instructed <u>and regularly drilled</u> as to the action to take at time of fire, first, toward assuring the safety of the men and, second, toward protecting property. Bosses and others familiar with the mine should be trained in the use of oxygen mine rescue apparatus and other similar equipment; and bosses, at least, should be familiar with mine-fire gases, the danger from them, and methods of detecting them. Above all other things, discipline of the strictest order should be maintained in all activities at or around a mine fire.

It is of extreme importance that material and apparatus on hand for use in case of fire be subject to rigid and efficient periodical inspection, and be given an occasional test under conditions approximating those of an emergency. It is exasperating, to say the least, to find that fire extinguishers are out of order, or hose spanners lost, or fire hose broken, or valve wheels displaced, after much dangerous effort has been made to get into position to fight a fire; and it is from delays caused by lack of proper attention to such details that many mine fires get out of hand. It is of

particular importance, not only at emergency work on a new fire, but at the more long-drawn-out task of controlling an established fire, that orders respecting the fire fighting be issued preferably by some one person, thus preventing confusion due to duplication of instructions.

The establishment of an efficient mechanical ventilation system, as previously mentioned, is of great value in fighting a fire, and mechanical ventilation is now placed at many metal mines where its chief use is to be in connection with possible fires. In addition to having a large fan to control main air currents, some mines are equipped with small auxiliary fans on mine trucks with a supply of flexible tubing and other accessories; this permits controlling the ventilation locally from strategic points near an underground fire. Some mining companies combine ventilation doors with fire doors, and erect concrete jambs with metal doors at points well located to isolate main air courses, if desirable; this permits controlling the flow of air as well as the fire or fire fumes. It is desirable that these fire doors be so constructed as to cause the door to close against the side of the concrete frame or jamb rather than against a groove <u>in</u> it, in order that settlement of the surrounding area will not prevent the opening or closing of the door. It is equally desirable that the door have a positive latch by which it may be opened from either side; this latch should be of such construction that it will not rust tight or otherwise become inoperative.

Each mine should have always available a supply of canvas fire hose, with nozzle, to fit surface or underground water pipe; possibly a small fan with canvas tubing, and at least a dozen 3-cell electric flash lights with plenty of batteries and a number of readily portable fire extinguishers. A large mine should have at least ten sets of up-to-date oxygen mine rescue apparatus, with supply of oxygen and regenerators and possibly other equipment. If gas masks (respiratory apparatus which filter out noxious gases but do not supply oxygen) are used in connection with metal-mine fires the greatest possible care should be exercised against bringing them into atmospheres which are or which may readily become deficient in oxygen or which contain poisonous gases in excess of the filtering capacity of the mask; and in any case no person should be allowed to use any type of respiratory apparatus such as oxygen breathing apparatus, or gas masks, or even hose masks unless he knows the details of the operation of the apparatus and its limitations and also has had instruction and at least some practice in the wearing of the apparatus. Small mines or mines at which oxygen apparatus may not be available can secure the apparatus as well as supplies and the advisory services of men accustomed to fighting fires by applying to the nearest U. S. Bureau of Mines safety office, station, or car. If its location is not known, quick action can be had by wiring for help to the Director of the United States Bureau of Mines, at Washington, D. C., who will promptly notify the nearest available safety car or office.

Metal-mine fires with heavy loss of life occur relatively seldom, and while much is written and read and said about fires for a few weeks after

occurrence of one which has caused heavy loss of life or possibly of property, in general mine fires and mine-fire data are given but scant attention between occurrence of fires. And when a mine fire occurs with unpleasant consequences it is almost invariably ascribed to an unavoidable chance or is called an act of Providence or of some other agency when as a matter of fact practically all mine fires are man-made and the result of gross carelessness. Moreover, their occurrence can be avoided, or at least loss of life in connection with them can be prevented, if every underground mine would frankly recognize that it has a fire hazard and would take measures such as are recommended in this paper for the avoidance of fires or for the safe and efficient handling of them if they should occur.

A relatively short bibliography is appended to this paper and from it can be selected much of the available metal-mine fire data published during the past several years.

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Department of Commerce

UNITED STATES BUREAU OF MINES
Scott Turner, Director

INFORMATION CIRCULAR

SUPPLEMENTARY NOTES ON CORE DRILLING IN THE SALT BEDS OF WESTERN TEXAS AND NEW MEXICO: TESTS 13 TO 24

BY

E. P. Hayes

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SUPPLEMENTARY NOTES ON CORE DRILLING IN THE SALT BEDS OF WESTERN TEXAS AND NEW MEXICO. TESTS 13 TO 24[1]

By E. P. Hayes[2]

INTRODUCTION

This report supplements the date contained in Bureau of Mines Information Circular 6156, "Special features of core drilling in the salt beds of Western Texas and New Mexico," by J. S. Wroth. That circular dealt with the drilling of the first 12 of the 24 potash tests made by the U. S. Bureau of Mines in cooperation with the U. S. Geological Survey; this report is concerned with the drilling of the last 12 tests.

ACKNOWLEDGMENTS

The writer is indebted to R. E. Heithecker, the Bureau of Mines engineer who had charge of the drilling of tests 13 to 17, for his well-prepared records on the drilling of these tests.

PRINCIPAL POTASH MINERALS PRESENT

Three potash minerals have been found in the Permian Basin in important quantities: (1) Polyhalite, a potassium, magnesium, and calcium sulphate ($2CaSO_4.MGSO_4.K_2SO_4.2H_2O$) which is very slowly soluble in water; (2) carnallite, a potassium, magnesium chloride ($KMgCl_3.6H_2O$) which is extremely soluble in water; and (3) sylvite, a potassium chloride (KCl) which is readily soluble, especially in hot water. It is obvious that if polyhalite is present it can always be discerned in well cuttings, while carnallite and sylvite, being more readily soluble in water, would nearly always dissolve in the water used for ordinary drilling.

DRILLING PRACTICES

Specifications.- Contract specifications are set forth in Information Circular 6156, previously referred to.

Equipment.- In addition to the description of equipment given in Information Circular 6156, further points of interest are noted in the following paragraphs.

Aside from requiring the use of a double core barrel, no specifications were made as to the equipment to be used. As it happened two contractors were successful in obtaining all the work. One contractor used steam for power, oil as fuel, a spudder to churn-drill the overburden, and a steam-powered core drill and steam-driven solution pump to core the salts. The other contractor used a gasoline-driven core drill, with fishtail bits to drill the over-

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6679."

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burden and standard diamond bits to core the salt. His solution pump was driven by a gasoline-powered tractor. One contractor used a steel derrick, the other a wooden A frame.

Each type of equipment has some advantages. A spudder may be used with advantage to drill in porous formations where the drilling solutions flow out of the hole, and the hauling of sufficient water to saturate the formation is too expensive. Abrasive formations such as certain types of sandstone wear the spudder bit but little, while fishtail bits have to be sharpened every 3 to 5 feet in such material, and in some cases more often. Fishing jobs on a spudder are usually less difficult than on a rotary machine, and less time is required to recover the bit. Fishtailing in caving formations has a decided advantage over the use of the spudder in that the drilling solution may be made heavy with mud, which cements the formations and tends to eliminate the caving. The water used in spudding has a decided tendency to make the formation softer and more apt to cave in and catch the spudding tools. With the use of the fishtail bit, more time is required to go in and come out of the hole, as the rods used must be unscrewed for each trip in and out, while the spudder drilling line winds on the reel without stopping. Three men are required as a crew for fish-tailing, whereas only two men are needed on a spudder rig.

The steel derrick furnishes a stronger structure than the A frame for the handling of casing and tools, more floor space to operate the drill, and being higher, a longer stand of rods can be used, which means a material saving of time in going in and coming out of the hole. The erection time is about the same for both rigs, although initial investment in the steel derrick is higher. A steel derrick may be used with either the spudding rig or with the fishtail rig.

Steam offers more flexible power and cheaper fuel than gasoline, but requires more water and more fuel. These are factors to be seriously considered in regions where water is scarce and where hauling of both water and fuel oil is difficult and expensive. Scale in the boiler must be contended with, and the boiler must be washed and cleaned about once a week to insure safe operation. Gasoline is a more expensive fuel than oil, but less of it is required and it is cheaper where haulage is considered. Gasoline-driven drills have greater maintenance costs than steam outfits, but the writer is of the opinion that in a country such as the Permian Basin, where practically all water contains much gypsum and where water and fuel must be hauled some distance, a gasoline driven rig is more cheaply operated. If a gasoline driven spudder is available, it is more desirable than a fishtail rig to drill the overburden.

Further Notes on Handling of Tests.- Procedure for drilling the overburden was left to the judgment of the contractor's drilling superintendent, and no definite program was laid out by the Bureau of Mines. Inasmuch as the responsibility for putting down the test lay with the superintendent, and the amount of casing used was of no interest to the bureau, he could start as large a hole as he thought necessary in order to get a full 2½-inch salt core, and could use just as much or just as little casing as he deemed wise, provided his casing shut off all water above the salt. The bureau engineer and the survey geologist tried always to give the contractor casing seats above the salt, so that he would not have to cement his last string of casing in the salt.

As a rule the drilling of the various tests proceeded along more or less the same lines. A hole large enough to take a joint of 10-inch casing was started at the surface; then the size of the hole was reduced and drilled to take 8¼-inch casing. The length of 8¼-inch casing varied, as in some tests it was used to shut off only the top water, and in others to shut off all waters above the salt. From the bottom of the 8¼-inch casing to the top of the salt, 6-5/8-inch casing was run, and landed or cemented as required to make a tight joint before starting to core the salt. Usually an attempt was made to find an anhydrite band on which to land this casing, but if this could not be done the casing was cemented in the next

best formation found. In several cases a tight seat was made by suspending the casing from the collar of the test and cementing it in the salt. Then 5 3/16-inch casing was run, since the 6-5/8-inch casing was too large for the pumps to lift out the cuttings from the hole, while the 5-3/16-inch gave approximately the proper velocity of the solution to clean the hole at a pressure at the bit which was not so high as to cause washing of the core. In case it was necessary to shut off leakage of the drilling solution in the salt, $4\frac{1}{2}$ or $3\frac{3}{4}$ inch flush-joint casing was run or the 5-3/16-inch casing was pulled, the hole reamed, and the 5-3/16-inch casing lowered. The flush-joint casing was too light in weight to be handled without great care and was used only as a last resort.

The drilling solution was made up in a large tank or sump by first saturating water with salt. Before starting the core drilling, the contractor was as a rule allowed to test the tightness of the hole by filling it with this saturated salt solution. No damage was done the salt formation in case the hole was not tight and the salt solution leaked out, and the salt solution which would be lost due to leakage was much cheaper than the solution saturated with salt, magnesium chloride, and potassium chloride. If the hole was found to be tight, 3 pounds of magnesium chloride and a $\frac{1}{4}$ pound of potassium chloride per gallon of saturated salt solution were then added and the solution was circulated through the hole for several hours. In this way the solution was thoroughly mixed and the solution originally used in the hole for testing was saturated with the other two chloride salts. With the addition of the magnesium chloride, some of the sodium chloride was precipitated and with the addition of potassium chloride, both sodium and magnesium chloride were precipitated to some extent. A sump placed in the return line or ditch between the drill hole and solution supply sump collected the cuttings and some of the surplus salts, though most of the precipitated salts were thrown out in the supply sump. As the hole was deepened and more solution became necessary, saturated solution was added in small quantities from a reserve supply tank. In the event that solution was lost for any reason whatever, the hole was cemented as outlined in a following paragraph, and new solution was made up and circulated as if the core drilling had just started.

Cementing in the salt to shut off leakage was successful if properly done. Too often, however, it was attempted without following the proper procedure. The following method has been found to be successful. The hole is bailed dry to a point below where the cement is to be placed, and a wooden plug fitting the hole tightly is set just below the point where the shut-off is desired. The cement is mixed in a box in the proportions of 5 gallons of water to one sack of cement, and an accelerator is added just before the mixture is run into the hole. The cement can either be poured through the rods, pumped in with the solution pump, or placed with a bailer. If rods are used care must be taken to see that they are not left in the cement long enough to become set in the cement. A second plug is run on to the top of the cement, a weight is placed on it to force the cement out into the crevice or point of leakage, and the hole is filled with water to hold the plug in place. After the cement is thoroughly set, the plug is drilled out and the hole tested to see that it is tight.

The amount of core cut during a shift depends upon the character of the formation and the skill of the driller. The driller was responsible for his shift, and was in charge of the operations, subject to orders from the drilling superintendent or tool pusher.

As a rule the weight of the rods and core barrel was sufficient for cutting any salt formation, sylvite, carnallite, clay, shale, or other soft formations. To cut the polyhalite or anhydrite, it was generally necessary to use the hydraulic feed to force the bit through these formations.

The use of diamonds was specified for core drilling in salt formations. Bit wear of diamonds was small, since very little abrasive material was encountered after the overburden was drilled, but diamonds had a tendency to tear the core, especially if they were well

rounded and well suited for drilling in harder rock. A set of well-rounded stones, specially selected from a large stock of excellent rock-cutting diamonds, was found to cut a very poor core. If new diamonds were used and set with the sharp, unworn edges out, they cut a good core but had a tendency to chip. Bortz was used successfully when set with the sharp edges out, and had an advantage over ordinary black diamonds in being less expensive; in the event that the bit was stuck or lost in the hole and could not be recovered, less money was lost. Bits made up with stellite teeth, or with teeth of similar especially hard alloy, were found to cut a more satisfactory core with less tearing than the diamonds. Two types of bits were made up and both proved to be satisfactory if the teeth were kept sharpened. The first bit was made by building up, with a welding torch, triangular-shaped teeth onto a regular bit blank. Twelve teeth were used, in four sets of three teeth each, the teeth being narrower than the face of the bit and staggered to allow free passage of solution at the cutting point. Each alternate tooth had a wing on the side of the bit to insure ample clearance between the bit and the side of the hole. The bit was set up in a lathe and was ground to gage both inside and outside with an emery wheel. The face of each tooth was then ground so that the cutting edge extended completely across the face of the tooth. The second type was made by setting sharpened stellite inserts in a regular bit in the same manner as diamonds are set in the bit. Both types should be set with an edge out and either changed when dulled or reground to keep sharp and to gage.

Most of the poor core was caused by etching or washing, between which it was difficult to distinguish; etching was due to an unsaturated drilling solution which dissolved the core, and washing to too high a solution pressure at the drill bit which caused a high velocity of the solution and wearing away of the salt. Etching usually showed in the sylvite or carnallite bands as a leaching out of spots in the core, while washing showed as vertical runs along the length of the core. In neither case was the core lost, but a fair determination of the potash in the core could not be made if any appreciable portion of the surface was etched or washed away.

Losses of core were due to several causes. The core barrel held 15 feet of core when full, and was usually pulled when it contained 13½ to 14 feet of core. If a driller took more than 15 feet of core, he ground up some of it to get it all in the barrel. If the core became blocked (that is, jammed or stuck in the inner barrel) some of it was bound to be ground up unless the driller pulled the core-barrel at once. In either case, some of the core was lost. Sometimes when the barrel was pulled, the core did not break exactly at the bottom of the hole and a stub of core was left. The length of the stub has been as much as 4 feet. If the driller was aware that the stub had been left and was careful in going back in the hole, he could set the barrel over the stub and pick it up with the next barrelful; otherwise it was crushed and lost. Practically all core losses were due to carelessness, though there were some extremely soft formations that washed away rather than cored.

If there was no loss of core, the length of the core as cut was always longer than the depth of the hole because the core broke at the edges of the various formations. It could not be fitted together as it was in the hole, and gave an overage. The driller got no credit for overage of core, but was charged with all losses. The overage was absorbed in the report made by the bureau engineer when he measured the core each week. Inasmuch as the contractor got a bonus for core recovered, it was to his advantage to bring up all the core possible. After the core was measured, the box was nailed or locked and was hauled to the Bureau of Mines core house in town, where it was held to be turned over to the survey's geologist for a careful geological logging and the cutting of the economic portions. These portions were sent to Washington for chemical analysis and final report by the U. S. Geological Survey.

The following tables give a complete record of operations at each well, including contract costs and average figures, for Wells 13 to 24, inclusive.

Performance on the Last 12 Tests

Well No.	Location County	Location State	Total depth Feet	Total depth Inches	Footage drilled in overburden Feet	Footage drilled in overburden Inches	Footage drilled in salts: Footage Feet	Footage Inches	Core-recovery Feet	Core-recovery Inches	Core-recovery, per cent
13	Eddy	N. Mex.	2,139	0	850	0	1,289	0	1,281	3	99.4
14	Lea	do.	[1]2,027	6	[1]1,179	6	848	0	822	0	96.9
15	Eddy	do.	1,079	6	514	0	565	6	565	6	100.0
16[2]	Chaves	do.	514	0	361	6	152	6	152	6	100.0
17	Lea	do.	[3]2,825	0	[3]1,565	0	1,260	0	1,260	0	100.0
18	Loving	Texas	1,876	0	1,205	0	671	0	669	0	99.7
19	Lea	N. Mex.	2,011	6	1,243	0	768	6	767	2	99.8
20	Lea	do.	2,302	4	1,325	0	977	4	977	4	100.0
21[4]	Lea	do.	3,010	0	2,079	0	931	0	925	0	99.4
22	Eddy	do.	1,724	0	606	0	1,118	0	1,103	0	98.6
23	Eddy	do.	904	0	495	0	409	0	406	0	99.3
24	Grand	Utah	1,731	0	1,178	0	553	0	540	6	97.7
Totals			[5]22,143	10	[5]12,601	0	9,542	10	9,469	3	99.2

1 - Test 14 was originally an oil and gas structure test drilled by the Humble Oil & Refining Co., it was taken over at 1,179 feet 6 inches and cored from that point.

2 - Shallowest test drilled by the Bureau of Mines.

3 - Test 17 was an oil and gas test drilled by the Empire Gas & Fuel Co.; it was plugged back to 1,540 feet, a deflector set in the hole, and cored from 1,565 feet where a full-sized core was first obtained.

4 - Deepest test drilled by the Bureau of Mines.

5 - Overburden of tests 14 and 17 included in this figure.

Casing Record and Contract Costs for Churn and Core Drilling, Wells 13 to 24

Well No.	Casing record: Set Feet	Set Inches	Recovered Feet	Recovered Inches	Contract Costs, Churn drilling: Total	Per foot
13	1,514	0	740	0	$4,887.50	$5.75
14	2,050	0	138	3	[1]2,398.51	2.03
15	897	0	897	0	2,955.50	5.75
16	252	0	212	0	2,078.63	5.75
17	4,588	0	3,038	9	[2]2,896.20	1.85

1 - Structure test taken over from Humble Oil & Refining Co. at 1,179 feet 6 inches. This amount represents amount paid drilling contractor for casing which was not recovered.

2 - Test was taken over from Empire Gas & Fuel Co. and plugged back to about 1,540 feet. A deflector was set and coring started. At 1,565 feet the first full-sized core was obtained. This amount represents the cost of deflecting the hole ($1,500) and the amount paid the Empire Co. for casing left in the hole ($1396.20).

Casing Record and Contract Costs for Churn and Core Drilling, Wells 13 to 24 - Continued

Well No.	Casing record: Set, Feet	Set, Inches	Recovered, Feet	Recovered, Inches	Contract Costs, Churn drilling: Total	Per foot
18	2,702	0	1,902	0	7,230.00	6.00
19	2,995	11	2,907	7	6,090.70	4.90
20	2,992	0	2,488	0	6,492.50	4.90
21	No record				12,370.05	5.95
22	860	6	716	6	3,605.70	5.95
23	726	0	475	0	2,945.25	5.95
24	1,202	5	101	0	7,009.10	5.95
Totals	20,779	10	13,616	1	60,959.64	4.84

Casing Record and Contract Costs for Churn and Core Drilling, Wells 13 to 24 - Continued

Well No.	Contract Costs: Core drilling: Drilling	Core recovery	Total	Per foot	Total Churn & core drilling	Per foot of total depth
13	$10,956.50	$1,281.25	$12,237.75	$9.49	$17,125.25	$8.01
14	7,208.00	822.00	8,030.00	9.47	10,428.51	5.14
15	4,806.75	565.50	5,372.25	9.50	8,327.75	7.71
16	1,296.25	152.50	1,448.75	9.50	3,527.38	6.86
17	11,970.00	2,520.00	14,490.00	11.50	17,386.20	6.15
18	6,374.50	1,338.00	7,712.50	11.49	14,942.50	7.96
19	5,917.45	1,534.32	7,451.77	9.70	13,542.47	6.73
20	7,525.44	1,954.67	9,480.11	9.70	15,972.61	6.94
21	9,310.00	1,850.00	11,160.00	11.99	23,530.05	7.82
22	11,181.00	2,206.00	13,387.00	11.97	16,992.70	9.86
23	4,090.00	812.00	4,902.00	11.99	7,847.25	8.68
24	5,530.00	1,081.00	6,611.00	11.95	13,620.10	7.87
Totals	86,165.89	16,117.24	102,283.13	10.72	163,240.77	7.37

Distribution of Time Consumed in Sinking Wells 13 to 24

Well No.	Total days on site	Moving in, setting up, plugging etc., days	Available for drilling		Actual drilling time		Miscellaneous delays	
			Days	Per cent	Days	Per cent	Days	Per cent
13	82	41	41	100	26.8	65.4	14.2	34.6
14	63	36	27	100	13.9	51.5	13.1	48.5
15	37.2	[1]17	20.2	100	10.5	52.0	9.7	48.0
16	22.7	10	12.7	100	6.4	50.4	6.3	49.6
17	62.4	[2]33	29.4	100	21.2	72.1	8.2	27.9
18	110.3	14	[3]96.3	100	31	32.2	[3]65.3	67.8
19	115.3	[4]38	[5]77.3	100	30.9	40	[5]46.4	60
20	88	[6]40	48	100	35.9	74.8	12.1	25.2
21	82	[7]26	56	100	42.7	76.3	13.3	23.7
22	55.1	22	33.1	100	22.2	67.1	10.9	32.9
23	27	11	[8]16	100	11.2	70	4.8	30
24	96.2	[9]12	[10]84.2	100	24	28.5	60.2	71.5
Totals	841.2	300	541.2	100	276.7	51.1	264.5	48.9

1 – Includes 6 holidays.
2 – Includes 6 days on deflecting job and 6 days waiting for deflector cement to set.
3 – Includes 9.3 days when no night crew was used and 39.6 days spent in fishing for diamond bit.
4 – Includes 13 Sundays that crews did not work.
5 – Includes 32.3 days spent in fishing for core bit.
6 – Includes 14 Sundays and holidays that crews did not work, also 7 days making water well for rancher.
7 – Includes 10 days taken by trucking contractor to move rig from Utah.
8 – Includes 3.3 days when night crew was not used.
9 – Does not include time used to truck equipment to Utah.
10– Includes time lost waiting for 9 cementing jobs to set (12 days); 2 days that crews did not work waiting for water; 5 days that crews did not work waiting for chloride; 20 days delay while attempting to cement with salt solution in hole; and 2 days during which well was shut down while solution was being made up to saturation point.

Data on Churn Drilling

Well No.	Date started	Date finished	Total time, days	Actual drilling time: Days	Actual drilling time: Per cent	Footage per day of— Total time	Footage per day of— Drilling time
13	July 23, 1929	August 8, 1929	16	11.7	73.1	53.1	72.6
14	No churn drilling		None	None	-	-	-
15	December 2, 1929	December 13, 1929	11	6.7	60.9	46.7	76.7
16	January 8, 1930	January 19, 1930	10.3	4.7	45.6	35.1	76.9
17	No churn drilling		None	None	-	-	-
18	May 29, 1930	July 9, 1930	40.5	19.5	48.2	29.8	61.8
19	July 24, 1930	August 16, 1930	[1]20.	14.9	74.5	62.1	83.4
20	November 20, 1930	December 23, 1930	[2]28.	18.7	66.8	47.3	70.8
21	September 11, 1931	October 15, 1931	34	24	70.6	61.1	86.6
22	May 28, 1931	June 14, 1931	16.1	8.8	54.7	37.6	68.9
23	May 13, 1931	May 24, 1931	11	6.2	56.4	45.0	79.8
24	July 28, 1931	September 4, 1931	38.4	15.7	40.9	30.7	75.0
		Totals	225.3	130.9	58.1	[3]43.7	[3]75.3

1 - Three Sundays are not included.

2 - Five Sundays are not included.

3 - Does not include footage on tests 14 and 17.

Data on Core Drilling

Well No.	Date started	Date finished	Total time, days	Actual drilling time: Days	Actual drilling time: Per cent	Footage per day of— Total time	Footage per day of— Drilling time
13	October 18, 1929	November 12, 1929	25	15.1	60.4	51.6	85.4
14	August 28, 1929	September 24, 1929	27	13.9	51.5	31.4	61.0
15	December 13, 1929	December 22, 1929	9.2	3.8	41.3	61.5	148.8
16	January 19, 1930	January 21, 1930	2.4	1.7	70.8	63.5	89.7
17	April 18, 1930	May 17, 1930	29.4	21.2	72.1	42.8	59.4
18	July 10, 1930	September 14, 1930	[1]55.8	11.5	20.6	12.0	58.3
19	August 21, 1930	October 28, 1930	[2]57.3	16	27.9	13.4	48.0
20	January 4, 1931	January 28, 1931	[3]20	17.2	86	48.9	56.8
21	November 11, 1931	December 3, 1931	22	18.7	85	42.3	49.8
22	June 20, 1931	July 10, 1931	[4]17	13.4	78.8	65.8	83.4
23	May 25, 1931	May 30, 1931	5	5	100	81.8	81.8
24	September 4, 1931	November 21, 1931	45.8	8.3	18.1	12.1	66.6
		Totals	315.9	145.8	46.1	30.2	65.4

1 - Includes 39.6 days spent in fishing for diamond bit.

2 - Ten Sundays are not included, but 32.3 days spent in fishing for diamond bit are included.

3 - Four holidays are not included.

4 - Three holidays are not included.

DRILLING NOTES ON INDIVIDUAL TESTS

Test 13.- Moving into the site covered eight days, a major part of which time was spent in building bridges and roads. Two days were spent in fishing for a bailer that was lost in the hole, but aside from that the spudding was done without delay. A gas pocket encountered while coring at 932 feet blew the solution from the hole. The gas soon exhausted itself, and it was not necessary to cement, although at a lower depth the solution was lost in a crevice and two cementing jobs were done before circulation was regained.

Plugging and dismantling consumed 13 days. The 6-5/8-inch casing was shot at various depths and attempts were made to pull it. It finally came loose at the top joint, the tightness probably being due to cavings holding the casing rather than cement.

Test 14.- A new drilling machine was sent in to core this test. Twenty-one days were required to rig up, as a number of parts had not been sent from the factory and had to be sent out later. Delays in drilling were accounted for mainly by one driller's being ill and losing 10 shifts of eight hours, and 4 idle days were caused by having to wait for fishing tools. Waiting for the cement to set on the first string of casing caused the balance of the delay.

Ten days were taken to plug the test and to dismantle the rig, practically all of which was due to trouble in pulling the casing. Only 160 feet of 2,050 feet of casing was recovered. This hole had been taken over from the Humble Oil & Refining Co. after they had completed a structure test to 1,181 feet and 6 inches. Government paid the Humble Co. contractor for the casing left in the hole. This amount ($2,398.51) represents the cost of drilling the overburden.

Test 15.- The 6-5/8-inch casing was set at 282 feet and the hole cored from 282 feet to 467 feet. The hole was then reamed to 513 feet and the 5-3/16-inch casing landed. Coring was resumed to 611 feet. At this point the solution was lost; so the 5-3/16-inch casing was pulled, the hole reamed to 611 feet 6 inches, and the 5-3/16-inch casing rerun. A day was lost when an air or gas pocket was drilled into at 603 feet, and drilling had to stop until the gas was exhausted.

No casing was lost, as the casing was not cemented and was easily pulled.

Test 16.- This test was drilled from top to bottom with the coring machine, a fishtail bit being used on the overburden and a core bit on the salt. Delay in drilling the overburden was accounted for by a 3-day wait for the cement to set in a crack at 344 feet, and a 2-day delay occasioned by extremely cold weather which froze the pumps. No night shift was used during the coring and there were no other delays.

Test 17.- Well 17 was originally drilled by the Empire Gas & Fuel Co. as an oil and gas test, was plugged back above the salt, a steel deflector cemented in the hole, and coring started. The deflector was especially made for the job and was set at 1,540 feet in the 6-5/8-inch casing with 20 sacks of cement. Coring was then started, the bit drilling through the side of the 6-5/8-inch casing and a full core being taken at 1,565 feet. Six days were required to set the deflector and an additional six days for the cement to set thoroughly.

Trouble was experienced in recovering the casing, and nearly all of it was left in the test. Eleven days were taken to plug the test and dismantle the rig. The Government paid $1,500 for the deflecting job and $1,396.20 for the casing left by the oil company in the test. This total ($2,896.20) is the amount that is chargeable to the drilling of the overburden.

Test 18.- Of the delays on drilling the overburden, about 10 days are chargeable to the fact that no night crew was used. A large part of the overburden was sandstone which wore out the fishtail bits after from 6 inches to 2 feet of drilling. The bits were faced with stellite, but though it was better than steel, the sandstone wore it very quickly. Over-

$1,500 was spent by the contractor for building up the stellite fishtail bits.

At 1,295 feet a diamond bit was twisted off the core barrel and remained in the hole when the driller started coring before he had proper solution circulation. The 5-3/16-inch casing had been dropped and the last joint badly bent so that no fishing tools could pass through the casing. When an attempt was made to pull the casing, it parted and left five joints in the hole. These five joints were drilled up with a special steel cutting bit and the diamond bit finally recovered. Nearly 40 days were lost in fishing for the bit and in clearing the hole of the steel chips which had dropped in while the casing was being drilled up.

Test 19.- No difficulties were encountered in cutting the overburden which was drilled with a spudding machine. Coring was started at 1,243 feet. At 1,306 feet the diamond bit became wedged in the hole in a dolomitic formation and the bit was twisted off. Left-hand rods were obtained from the factory and the drilling rods were backed off the core barrel, together with the water head and inner core barrel. A special reamer was then made and the outer core barrel was drilled up. A chip box was used on the reamer and all the steel chips were recovered from the hole. The chips were weighed and found to be within 2 ounces of the weight of the outer core barrel when new. When the outer core barrel had been cut to the diamond bit and the hole washed for 24 hours with salt solution, the bit was caught on a taper tap and taken from the hole. Thirty-two days were lost in waiting for the left-hand rods and in fishing for the bit.

Twelve days were used in plugging test and dismantling rig. The rig was moved to the next location as it was dismantled. A sandy trail caused the work to proceed slowly. Very little casing was lost, as only the 5-3/16-inch was cemented.

Test 20.- No trouble was met with in drilling the overburden. At 1,125 feet a band of salt was found and without waiting for approval, the contractor started to tear down his spudder and erect his core drill. When he found that some dolomitic formations were still to be drilled and that he was apt to stick his diamond bit again, he tore down the core drill and erected his spudder to drill about 200 feet more. About five days were lost on this account.

The contractor needed additional casing to finish the hole and ran a string of extremely light 3-3/4 inch, some of which was lost when he pulled the casing. An extra seven days were consumed when the completed test was converted to a water well for a rancher.

Test 21.- The drilling of the overburden was done on subcontract by a local well-drilling contractor, who started too small a hole in order to save a string of casing. At 1,076 feet a quicksand was struck, and although the contractor tried for two months he was not able to drill deeper. As he was short a string of casing, he could not case off the quicksand and finally had to abandon the hole. A new subcontractor who had had experience in the locality drilled the overburden with very little delay, and any that occurred was due to mechanical troubles. Coring was completed without serious delay.

Test 22.- The overburden drilling was done by a subcontractor, and no record was kept by him of any special features. Some trouble was met on account of lack of water because of the extreme drought at the time and the subcontractor was required to haul his water from considerable distances.

Two days were spent in covering the trails with bear grass in order that the trucks hauling the coring machine could pass over them to reach the location. Three days were lost when the crews took time off around the Fourth of July.

As the coring machine was sent to U ah to drill test 24, eight days were taken to plug the test, dismantle the rig, and load the equipment on trucks.

Test 23.- The overburden was drilled with the coring machine, using a fishtail bit. No night crew was used and three days delay was charged to the total time available for drilling

on that account. Two gas pockets blew the solutions from the hole at different times but no time was lost.

Test 24.- Test 24 was drilled in Utah; it was the only test drilled outside of Texas or New Mexico and at a considerable distance from the base of supplies. The overburden was sandstone, badly broken, and very close to a fault. Rock bits solved the drilling, but crevices required nine cementing jobs, which caused a total delay of 12 days. Water had to be obtained from a seep, and four days were lost while the crews waited for enough water to flow into the seep so that they could have enough to drill continuously.

Coring was started at 1,178 feet. At 1,206 feet, a leak in the formation developed and an attempt was made to cement without bailing the salt water from the hole. Later a second attempt was made at the same point without success. Finally the hole was bailed, a plug set below the leak, the cement run in through the rods, and a second plug set on the cement. After three days the plug was drilled out and the test completed without any further difficulty. Twenty days were lost on these three cementing jobs that could have been saved. Seven days were lost on account of not having enough magnesium chloride on hand, and having to wait while trucks brought 12 tons from the contractor's warehouse at Carlsbad, N. Mex. The equipment was then trucked to New Mexico and the twenty-first test was cored. This completed the work.

ACCOPRESS BINDER

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